TRANSACTIONS

OF THE

AMERICAN INSTITUTE OF MINING AND METALLURGICAL ENGINEERS

(INCORPORATED)

1929

YEAR BOOK

THIS VOLUME CONTAINS PAPERS AND DISCUSSIONS PRESENTED AT THE NEW YORK MEETING, FEBRUARY, 1929; OFFICERS, COMMITTEES AND LIST OF MEMBERS

NEW YORK, N. Y.
PUBLISHED BY THE INSTITUTE
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1929

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PREFACE

This volume of Transactions is the fourth issued by the Institute for 1929 and completes the series for the year. The preceding volumes were

Transactions, Petroleum Development and Technology, 1928–29
Transactions, Institute of Metals Division, 1929.
Transactions, Iron and Steel Division, 1929.

The present volume includes the reports on the activities of the Institute throughout the year, together with a complete index to all material published in any form. In the index there are direct references to abstracts printed in Mining and Metallurgy of all papers as issued. By use of the index and the abstracts the work of looking up anything published by the Institute in 1929 will be found to be easy. With this volume, also, is the List of Members, previously printed each spring, but in 1929 held back so that all members might find in one volume, bound for permanent preservation, a complete record for the year. The addresses have been corrected up to December, 1929, so far as members have supplied the data.

The technical papers and discussions in this volume cover the related groups of metal mining, mining geology, non-ferrous metallurgy and non-metallic minerals. Those relating to petroleum, iron and steel, and the activities of the Institute of Metals Division appeared in the three Transactions volumes already printed and distributed. Papers on coal, geophysics and milling are held over in expectation that in 1930 or later enough material on each subject will have accumulated to warrant a special volume devoted to each. It will be recalled that volumes on Flotation Practice and Geophysical Prospecting were issued in 1928 and 1929, respectively, before the present plan of Transactions volumes was adopted.

All Transactions volumes are now furnished bound in cloth without extra charge. Those who had paid for the former standard binding will receive this volume so bound. If sufficient demand exists future volumes will also be made available in that form.

The present volume is being sent without request to all members and associates in good standing. In addition each member who had on file a request to that effect has received that one of the other Transactions volumes which related to the subject of chief interest to him. The other volumes will be furnished members on request at a cost of $2.50

each. Thus each member is entitled to this and one other volume bound, without charge. He will be expected to pay the cost of any additional volumes.

Members and associates will continue as heretofore to receive free any or all of the unbound *Technical Publications*, which series includes the larger part of the pamphlets issued each year. These may be asked for as a whole, by series, or individually. A notice and an abstract of each appears in MINING AND METALLURGY when it is issued. Members are free to change at any time their listing for receipt of such publications. The special groups recognized in this series are as below:

A—Metal Mining
B—Milling and Concentration
C—Iron and Steel Division
D—Non-ferrous Metallurgy
(Reduction of Ores)
E—Institute of Metals Division
(Founding and Metallography of Non-ferrous Alloys)
F—Coal and Coke
G—Petroleum Division
H—Non-metallic Minerals
I—Mining Geology
K—Mining Administration
(Accounting, taxation, industrial relations)
L—Geophysical Prospecting

MINING AND METALLURGY will be sent to all members each month, as heretofore; by reading it each member can keep in touch with the Institute, its meetings, and its publications. By watching the abstracts there a member can keep up to date on a wide variety of subjects with a minimum of time.

On page 357 will be found a complete list of pamphlets published within the year. In all 3130 unduplicated pages of printed matter (exclusive of MINING AND METALLURGY) have been issued for the use of Institute members. Publication absorbs 50 per cent. of the funds of the Institute, an expenditure justified by the variety and quality of the technical material made available to its members and the profession in general.

H. FOSTER BAIN.
Secretary.

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G. E. Johnson
Archibald Jones
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Wadsworth W. Norton
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Kurt Stock
George C. Stone
Otto Sussman
U. C. Tainton
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C. R. Kuzell
Gerald Sherman

Boston

Meets first Monday each winter month

Holcombe J. Brown, *Chairman*
Donald H. McLaughlin, *Vice-Chairman*
F. Leroy Foster, *Secretary-Treasurer*,
Room 8–219, Mass. Inst. of Tech., Cambridge, Mass.
Alfred C. Lane
George A. Packard

Chicago

Meets each month, September to May, inclusive

Wilfred Sykes, *Chairman*
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R. D. Crawford
R. H. Gordon
H. A. Stewart

Columbia

Holds noon meetings first Friday of each month. Annual meeting first Friday, December

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I. L. Box, *Treasurer*, Cons. Lead & Zinc Co., Picher, Okla.
P. B. Butler
C. F. Dike
Carl K. Carmean
W. F. Netzeband

Lehigh Valley

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H. B. Goodrich
Earle S. Porter
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C. J. Calvin, *Vice-Chairman*
S. E. Atkins, *Secretary-Treasurer*, Alworth Bldg., Duluth, Minn.

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Edmund S. Leaver
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Holds monthly meetings, except June, July and August

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A. B. Parsons, *Treasurer*
L. F. Strobel, *Secretary*, Room 5002, 233 Broadway, New York, N. Y.
William F. Boericke
A. L. J. Queneau
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Chester F. Lee, *Secretary-Treasurer*, 602 Colman Block, Seattle, Wash.
Thomas Davis
B. H. Bennetts

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Franklin B. Richards, *Chairman*
H. M. Boylston, *Vice-Chairman*
C. B. Murray, *Vice-Chairman*
Sidney K. Reid, *Secretary-Treasurer*, 1208 Hanna Bldg., Cleveland, Ohio
Harry E. Nold
C. Snelling Robinson

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Frank Whalley Watson, *Vice-Chairman*
J. H. Batcheller, *Secretary-Treasurer*, 963 Adams St., Corvallis Ore.
Albert Burch
W. W. Elmer

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W. J. Richards, *Vice-Chairman*
J. B. Warriner, *Vice-Chairman*
W. W. Inglis, *Vice-Chairman*
Paul Sterling, *Secretary-Treasurer*, Lehigh Valley Coal Co., Wilkes-Barre, Pa.
Charles Dorrance
G. B. Hadesty
A. B. Jessup
John M. Humphrey

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R. E. Davis
H. N. Eavenson
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T. M. Girdler
John G. Ryan

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Meets second Tuesday of each month

E. A. Hersam, *Chairman*
R. G. Hall, *Vice-Chairman*
James G. Parmelee, *Secretary-Treasurer*, 1022 Crocker Bldg., San Francisco, Cal.
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Van Court Warren, *Secretary-Treasurer*, 812 Subway Terminal Bldg., Los Angeles, Cal.
H. Norton Johnson
Robert R. Boyd
F. L. Ransome
Harry J. Morgan

Southern West Virginia and Northeastern Kentucky

J. K. Anderson, *Chairman*
Charles E. Krebs, *Secretary-Treasurer*, 2307 Washington St., Charleston, West Va.

St. Louis

Carl Stifel, *Chairman*
Charles L. Dake, *Vice-Chairman*
Alfred C. Callen, *Vice-Chairman*
John T. Fuller, *Vice-Chairman*
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H. O. Davidson
Lucien Eaton
Rudolph Ericson
A. W. Hopkins
W. H. Schacht

Utah

Meets second Thursday of each month, October to May, inclusive
J. A. Norden, *Chairman*
Robert Wallace, *Vice-Chairman*
C. T. Van Winkle, *Secretary-Treasurer*, Dooly Block, Salt Lake City, Utah

Washington, D. C.

George K. Burgess, *Chairman*
Howard I. Smith, *Vice-Chairman*
Edward B. Mathews, *Vice-Chairman*
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AFFILIATED STUDENT SOCIETIES

1. Any society of undergraduates at a technical school, comprising students in any branch of engineering, metallurgy, chemistry, geology, etc., may be recognized by the Board of Directors in its discretion as an Affiliated Student Society, and its name, together with the names of its president and its secretary, may be published in MINING AND METALLURGY or the DIRECTORY of the Institute. The members of such a society will not be named individually on the membership list of the Institute, but may at any time be proposed for election as Student Associates or Junior Members of the Institute, in the usual way.

2. Membership in an Affiliated Student Society, certified by the official representative thereof, and accompanied by suitable testimony as to character and qualifications, will have special weight with the Membership Committee of the Institute, in its consideration of the proposal of a candidate for membership.

The Affiliated Student Societies receive the following privileges:

1. A copy of MINING AND METALLURGY free;
2. The use of Employment Service;
3. The use of MINING AND METALLURGY for publicity purposes, such as printing notices of their activities, etc.;
4. Assistance in obtaining speakers for their meetings, lists of moving-picture films, etc.

To retain the privileges of the Affiliated Student Society, its officers must keep in touch with the Secretary of the Institute and advise him of its activities.

The foregoing plan is not confined to students in mining engineering or metallurgy. Any undergraduate society of engineers may avail itself of the privileges offered.

Societies desiring to be recognized as Affiliated Student Societies are requested to make application to the Secretary of the Institute, giving particulars as to the nature, membership and officers.

The following list gives the names of the Affiliated Student Societies as of December 1, 1929, and their presidents and secretaries.

Affiliated Student Societies

University or College	Society	President	Secretary
Alabama, University of, University, Ala	Mining and Metallurgical Society	Warren G. Payne	Melbourne J. Langley
Alaska Agr. College and Sch. of Mines, College, Alaska	Alaska Mining Society	Larry C. Doheny	John R. Wilcox
Arizona, University of, Tuscon, Ariz	Miners Society	D. C. Minton	Frank Bacon
California, University of, Berkeley, Cal	Students Mining Association	H. K. Lancaster	James Mullin
Carnegie Inst. of Technology, Pittsburgh, Pa	Mining and Metallurgical Society	Wilbur H. Seifert	George F. Mart
Case School of Applied Science, Cleveland, Ohio	Pick and Shovel Club	W. E. Sicha	H. E. A. Dorer
Harvard University, Cambridge, Mass	Geological Club	H. J. Fraser	P. H. Schafer
Idaho, University of, Moscow, Idaho	Associated Miners	Floyd E. Albertson	John Nicholson
Illinois, University of, Urbana, Ill	Mining Society	J. A. Bottomley	A. L. Barrett
Kansas, University of, Lawrence, Kan	Student Branch, A. I. M. E.	Henry Gould	Everett Stidham
Kentucky, University of, Lexington, Ky	Norwood Min. & Met. Society	Cresap Moss	Bernard Heafling
Lafayette College, Easton, Pa	John Markle Mining Society	E. H. Boos	K. H. Armstrong
Lawrence College, Appleton, Wis	Geological Engineers Club	Herbert Ramlowe	Malcombe Campbell
Lehigh University, Bethlehem, Pa	Mining and Geological Society	Kenneth R. Eckrote	Andrew M. Henry
Massachusetts Inst. of Technology, Cambridge, Mass	Mining Engineering Society	Robert Henderson	Robert S. Backus
Michigan College of Mines, Houghton, Mich	Ingot Club	R. B. Kropf	B. J. Werkowski
Michigan College of Mines, Houghton, Mich	Students Organization	Charles Bramer	Millard D. Crowell
Minnesota, University of, Minneapolis, Minn	School of Mines Society	M. W. Griswold	G. M. Carlson
Missouri School of Mines, Rolla, Mo	Missouri Min. & Met. Assn.	J. R. Jarboe	J. W. Graybeal
Nevada, University of, Reno, Nev	Crucible Club	R. W. Prince	H. I. Ashby
New Mexico State School of Mines, Socorro, N. M	Cooney Mining Club	Frank Stevens	T. P. Wootten
North Dakota, University of, Grand Forks, N. D	Mining, Industrial and Chemical Engineering Society	Arnott J. Lee	Leslie R. Else
North Georgia Agricultural College, Dahlonega, Ga	Mining Club	Andrew Hutchens	Richard S. McConnell
Ohio State University, Columbus, Ohio	Student Branch, A. I. M. E.	W. W. Ballantine	F. C. Hart
Oklahoma, University of, Norman, Okla	Pick and Hammer Club	Pat Darnay	Sam Zimmerman
Oregon State Agricultural College, Corvallis, Ore	Miners Club, School of Mines	Juel G. Huseby	Paul Aubert
Pennsylvania State College, State College, Pa	The Mining Society	F. T. Moyer	O. I. Savenius
Pittsburgh, University of, Pittsburgh, Pa	The Mining Society	E. J. Gallagher	H. S. Gibbs
Princeton University, Princeton, N. J	Princeton Engineering Society	John G. Myers	Robert M. Schafer
Stanford University, Stanford University, Cal	Geological and Mining Society	Alden B. Greninger	Charles M. Cross
Texas Agr. & Mechanical College, College Sta., Tex	The Geology Club	M. T. Halbouty	F. K. Peyton
Texas, University of, El Paso, Tex	The Scientific Society	E. B. Daggett	Alton Dunaway
Toronto, University of, Toronto, Canada	Mining and Metallurgical Club	Bertrand Robinson	J. H. E. Doyle
Tufts College, Tufts College, Mass	Tufts Chemical Society	C. V. Mann	H. F. Russell
Utah, University of, Salt Lake City, Utah	Utah Mining and Metallurgical Society	H. W. St. Claire	Walter Bates
Virginia Polytechnic Institute, Blacksburg, Va	Mineral Club	C. H. Hagy	C. Findlay
Washington, State College of, Pullman, Wash	Mining and Geological Society	Howard Woodham	John Hill
Washington, University of, Seattle, Wash	The Mines Society	D. A. P. Pifer	Malcolm Jack
West Virginia, University of, Morgantown, W. Va	Mining Society	Robert Baird Ryland	William A. Haslam
Wisconsin, University of, Madison, Wis	Mining Club	Theodore D. Tieman	Philip McCaffery
Yale University, New Haven, Conn	Mining and Metallurgical Society	W. R. Jennings	D. C. Jillson

INSTITUTE MEDALS AND PRIZES

Aside from the John Fritz Medal, in which the Institute participates through its representation on the John Fritz Medal Board, the Institute itself has four awards it may make annually, as follows: the Robert W. Hunt Medal and Prize, the J. E. Johnson, Jr., Award, the James Douglas Medal, and the William Lawrence Saunders Medal.

Robert W. Hunt Medal and Prize

The partners of Robert W. Hunt established a fund which was presented to the Institute at a fitting ceremony on May 27, 1920, to establish an annual medal and a sum of money to be awarded under the following rules:

1. The award shall consist of two prizes: first of a gold medal, second of a sum of money; a certificate to accompany each prize. The money prize shall not be awarded to a member over 40 years of age, but under unusual circumstances, both prizes may be allotted to one person provided that he is not over 40 years of age. In general it will be understood that the Committee shall award the money prize to the younger men, rather than the medal.
2. The awards shall be made not oftener than once a year to that person or persons contributing to the Institute the best original paper or papers on iron and steel. The scope of the term "iron and steel" shall be determined by the subcommittee considering the awards. In general, papers dealing with the practical side of the subject have preference over those dealing with the theoretical side in recognition of the fact that Captain Hunt's main contributions to the industry have been in the improvement of production and quality of material.
3. A subcommittee of three to five, including the Chairman of the Iron and Steel Committee, shall be appointed by the Iron and Steel Committee annually to adjudge the award subject to approval.
4. The awarding committee shall submit its report to the Iron and Steel Committee at its October meeting, and the award shall be certified to the Secretary of the Institute in time to permit the presentation to be made at the Annual Meeting of the Institute.
5. The recipient of the award shall be designated "The Hunt Medallist."

This prize is administered by the Iron and Steel Division.

Awards of the medal have been made as follows:

1920—Robert Woolston Hunt.
1926—Charles Lewis Kinney, Jr.
1928—John A. Mathews.
1929—Edgar Collins Bain.

J. E. Johnson, Jr., Award

This award is made from the income of a fund of $3000 donated by Mrs. Margaret Hilles Johnson in memory of her husband, J. E. Johnson, Jr., who was a prominent engineer, author of two valuable volumes on iron blast-furnace construction and practice, vice-chairman of the Institute's Iron and Steel Committee, and a frequent contributor of papers to the Institute's TRANSACTIONS.

This prize is administered by the Iron and Steel Division.

The intent of the donor is to encourage young men in creative work in branches of the metallurgy or manufacture of pig iron with which the professional activities of Mr. Johnson were chiefly concerned. The control of the fund and distribution of the awards having been vested in the Board of Directors of the Institute, this Board has made the following regulations concerning it:

(1) The award shall not be made to persons over forty years of age.

(2) The award shall not be made oftener than once a year.

(3) The award shall be accompanied by a certificate suitable for framing.

(4) The award shall be made to some promising engineer, selected because of a meritorious research, invention, or contribution to the professional literature of iron and steel, along the lines with which the professional activities of Mr. J. E. Johnson, Jr., were chiefly concerned.

(5) The Iron and Steel Division of the Institute shall recommend to the Board of Directors before the Annual Meeting of the Institute, each year, any suitable person qualified to receive the award.

(6) The Board of Directors of the Institute may change these regulations as may be necessary or advisable, within the spirit and intent of the donor's desire to bring encouragement and distinction to promising young engineers.

This prize has been awarded as follows:

1923—Alexander L. Feild.
1926—Selwyne Perez Kinney.
1927—Thomas L. Joseph.
1928—P. H. Royster.
1929—No award.

THE JAMES DOUGLAS MEDAL

Endowment for an annual gold medal, commemorating Dr. James Douglas, twice President of the Institute, was established by anonymous donors in 1922. The rules governing the award of the medal, as adopted by the Board of Directors, include the following main provisions:

1. The medal to be awarded once a year, preferably at the time of the annual meeting. The medal is to be accompanied in every case by an engrossed diploma containing a citation of the service or achievement upon which the award is based.

2. Eligibility. (*a*) Distinguished achievement in non-ferrous metallurgy. (*b*) No limitations as to nationality, membership in the Institute or otherwise. (*c*) Recipient must be a living person, able and willing to present himself in person to receive the award at the time and place to be prescribed by the Board of Directors.

3. The choice of a recipient is to be based upon a report by a committee of fifteen members of the Institute interested in non-ferrous metallurgy, chosen by the Board of Directors and distributed as widely as possible geographically. The tenure of office shall be for three years, so arranged that a portion of the Committee shall be elected each year. A member shall not be eligible for reappointment to the Committee for a period of one year after his term has expired. The President shall be ex-officio a member of the Committee but not its Chairman. The award shall finally

be made by a majority vote of the Board of Directors, after a full consideration of the report by the Committee. The Directors shall have the right to reject the report of the Committee.

4. The Committee may call on the membership for nominations, but such nominations must be accompanied by a detailed argument in favor of the award and details sufficient to permit a proper citation to be drawn.

Awards of the medal have been made as follows:

1923—Frederick Laist.
1924—Charles Washington Merrill.
1925—William Hastings Bassett.
1926—John Michael Callow.
1927—Zay Jeffries.
1928—Selwyn G. Blaylock.
1929—Paul Dyer Merica.

The William Lawrence Saunders Gold Medal

The William Lawrence Saunders Gold Medal for achievement in mining was established in 1927. The rules adopted for the award of this medal are as follows:

1. The Mining Medal may be awarded once a year, but not oftener. The medal is to be accompanied by an engraved diploma containing a citation of the achievement on which the award is based.

2. The presentation ceremonies will take place preferably during the Annual Meeting of the Institute, although the Directors may arrange for its presentation elsewhere.

3. The medal will be awarded for distinguished achievement in mining. There are no limitations as to nationality, membership in our Institute or otherwise.

The recipient must be a living person, able and willing to present himself in person to receive the award at the time and place prescribed by the Board of Directors.

4. The choice of a medallist is to be based upon a report by a Mining Medal Committee of fifteen members of the Institute interested in mining, chosen by the Board of Directors on the nomination of the President, and distributed as widely geographically as convenient. The tenure of office shall be for three years, so arranged that a portion of the Committee shall be elected each year. A member shall not be eligible for reappointment to the Committee for a period of one year after his term has expired. The President of the Institute shall, ex-officio, also be a member of the Committee, but not its chairman.

The award shall finally be made by a majority vote of the Board of Directors of the Institute after consideration of the report of the Committee. The Directors shall have the right to reject the report of the Committee.

5. The medal committee may invite nominations from the membership of the Institute, such nominations to be accompanied by an argument in favor of the award and details sufficient for a proper citation to be drawn.

This medal has been awarded as follows:

1927—David William Brunton.
1928—Herbert Hoover.
1929—John Hays Hammond.

SEELEY W. MUDD MEMORIAL FUND

This fund was established in 1929 for support of research and especially refers to non-ferrous mining and metallurgy. It is administered under the Board by a committee consisting of the President and Secretary and three members selected by the Board.

HOWE MEMORIAL LECTURE

The Howe Memorial Lecture, in memory of Henry Marion Howe, Past President of the Institute, was authorized in April, 1923, as an annual address to be delivered by invitation under the auspices of the Institute by an individual of recognized and outstanding attainment in the science and practice of iron and steel metallurgy or metallography, chosen by the Board of Directors upon recommendation of the Iron and Steel Division. The following were the Howe lecturers for the years indicated:

Albert Sauveur—1924
John A. Mathews—1925
William Campbell—1926
Bradley Stoughton—1927
Henry D. Hibbard—1928
John Howe Hall—1929

JOINT ACTIVITIES

The Institute conducts jointly with the American Society of Civil Engineers, American Society of Mechanical Engineers, and American Institute of Electrical Engineers, certain activities as listed below, and is joint owner with them of certain properties.

United Engineering Society

This is a corporation organized by the American Institute of Mining Engineers, American Society of Mechanical Engineers, and the American Institute of Electrical Engineers, to hold the title and administer the property at 29 West 39th St. In 1916, the American Society of Civil Engineers was admitted, and each society now has an equal one-fourth interest in the ownership, occupancy, and administration of the building. The cost of this property has been over two million dollars. A. I. M. E. representatives are: Arthur S. Dwight, John V. N. Dorr and H. A. Guess.

Alfred D. Flinn, Secretary, 29 West 39th St., New York City.

Engineering Foundation

This is a fund established by the United Engineering Society in 1915 with a gift of $200,000 from Ambrose Swasey. Subsequent gifts had increased the fund to about $620,000 on March 1, 1929.

The income of the fund is used to further research in science and engineering and to advance the profession of engineering and the good of mankind. A research project in alloys of iron is being conducted in close cooperation with the Institute. A. I. M. E. representatives are: Galen H. Clevenger, John M. Callow and John V. N. Dorr.

ALFRED D. FLINN, Secretary, 29 West 39th St., New York City.

JOHN FRITZ MEDAL BOARD OF AWARD

The medal was established in 1902 and has since been awarded not oftener than annually by a board of sixteen members, four representatives each from the American Societies of Civil, Mining and Metallurgical, Mechanical, and Electrical Engineers. This medal is awarded "for notable scientific or industrial achievement," and there is no restriction on account of nationality or sex. The board meets annually on the third Friday of October. For 1929, the Chairman is Bancroft Gherardi; Treasurer, George S. Davison; Secretary, Lincoln Bush; Assistant Secretary, Alfred D. Flinn, 29 West 39th St., New York.

JOINT CONFERENCE COMMITTEE

This is a committee consisting of the presidents and secretaries of the four National Engineering Societies, which meets at intervals to consider matters of mutual concern to the four societies and makes, where possible recommendations for concurrent action by the societies.

ENGINEERING SOCIETIES LIBRARY

This is a consolidation of the four libraries formerly maintained by the four National Engineering Societies and is administered through the Engineering Societies Library Board. It contains over 122,000 volumes and is conducted as a free public reference library. It is open to the public from 9 a.m. to 10 p.m. on week days, excepting July and August, when the library closes at 5 p.m. A. I. M. E. representatives are: Sydney H. Ball, George C. Stone, John H. Janeway and Joseph E. Pogue.

HARRISON W. CRAVER, Director, 29 West 39th St., New York City.

ENGINEERING SOCIETIES EMPLOYMENT SERVICE

A cooperative service for engineers and their employers under the direction of the American Institute of Mining and Metallurgical Engineers; American Society of Civil Engineers; American Society of Mechanical Engineers; and American Institute of Electrical Engineers, and with the cooperation in Chicago of the Western Society of Engineers, and in San Francisco of the California Section of the American Chemical Society and the Engineers' Club of San Francisco. For information as to men available or positions open, address the nearest office: *New York*, Walter V. Brown, *Manager*, 31 West 39th St., Telephone Penn. 9220; *Chicago*, A. Krauser, *Manager*, 1216 Engineering Bldg., 205 West Wacker Drive, Telephone Harrison 1238; *San Francisco*, Newton D. Cook, *Manager*, Room 715, 57 Post St., Telephone Sutter 1684.

LIST OF THE MEETINGS OF THE INSTITUTE AND THEIR LOCALITIES FROM ITS ORGANIZATION TO

DECEMBER, 1929

			Trans.	
No.	Place	Date	Vol.	Page
1.	Wilkes-Barre, Pa.*	May, '71	1	3
2.	Bethlehem, Pa	Aug., '71	1	10
3.	Troy, N. Y	Nov., '71	1	13
4.	Philadelphia, Pa	Feb., '72	1	17
5.	New York, N. Y.*	May, '72	1	20
6.	Pittsburgh, Pa	Oct., '72	1	25
7.	Boston, Mass	Feb., '73	1	28
8.	Philadelphia, Pa.*	May, '73	2	3
9.	Easton, Pa	Oct., '73	2	7
10.	New York, N. Y	Feb., '74	2	11
11.	St. Louis, Mo.*	May, '74	3	3
12.	Hazleton, Pa	Oct., '74	3	8
13.	New Haven, Conn	Feb., '75	3	15
14.	Dover, N. J.*	May, '75	4	3
15.	Cleveland, O	Oct., '75	4	9
16.	Washington, D. C	Feb., '76	4	18
17.	Philadelphia, Pa.†	June, '76	5	3
18.	Philadelphia, Pa	Oct., '76	5	19
19.	New York, N. Y	Feb., '77	5	27
20.	Wilkes-Barre, Pa.*	May, '77	6	3
21.	Amenia, N. Y	Oct., '77	6	10
22.	Philadelphia, Pa	Feb., '78	6	18
23.	Chattanooga, Tenn.*	May, '78	7	3
24.	Lake George, N. Y	Oct., '78	7	103
25.	Baltimore, Md.*	Feb., '79	7	217
26.	Pittsburgh, Pa	May, '79	8	3
27.	Montreal, Canada	Sept., '79	8	121
28.	New York, N. Y.*	Feb., '80	8	275
29.	Lake Superior, Mich	Aug., '80	9	1
30.	Philadelphia, Pa.*	Feb., '81	9	275
31.	Staunton, Va	May, '81	10	1
32.	Harrisburg, Pa	Oct., '81	10	119
33.	Washington, D. C.*	Feb., '82	10	225
34.	Denver, Colo	Aug., '82	11	1
35.	Boston, Mass.*	Feb., '83	11	217
36.	Roanoke, Va	June, '83	12	3
37.	Troy, N. Y	Oct., '83	12	175
38.	Cincinnati, O.*	Feb., '84	12	447
39.	Chicago, Ill	May, '84	13	1
40.	Philadelphia, Pa	Sept., '84	13	285
41.	New York, N. Y.*	Feb., '85	13	585
42.	Chattanooga, Tenn	May, '85	14	1
43.	Halifax, N. S	Sept., '85	14	307
44.	Pittsburgh, Pa.*	Feb., '86	14	587
45.	Bethlehem, Pa	May, '86	15	lxiii.
46.	St. Louis, Mo	Oct., '86	15	lxx.
47.	Scranton, Pa.*	Feb., '87	15	lxxvi.
48.	Utah and Montana	July, '87	16	xvii.
49.	Duluth, Minn	July, '87	16	xxiv.
50.	Boston, Mass.*	Feb., '88	16	xxviii.
51.	Birmingham, Ala	May, '88	17	xix.
52.	Buffalo, N. Y	Oct., '88	17	xxiv.
53.	New York, N. Y.*	Feb., '89	17	xxxi.
54.	Colorado	June, '89	18	xvii.
55.	Ottawa, Canada	Oct., '89	18	xxiv.
56.	Washington, D. C.*	Feb., '90	18	xxx.
57.	New York, N. Y	Sept., '90	19	vii.
58.	New York, N. Y.*	Feb., '91	19	xxv.
59.	Cleveland, O	June, '91	20	xvi.
60.	Glen Summit, Pa	Oct., '91	20	lxi.
61.	Baltimore, Md.*	Feb., '92	21	xix.
62.	Plattsburg, N. Y	June, '92	21	xxxiii.
63.	Reading, Pa	Oct., '92	21	xliv.
64.	Montreal, Canada*	Feb., '93	21	lii.
65.	Chicago, Ill	Aug., '93	22	xiii.
66.	Virginia Beach, Va.*	Feb., '94	24	xvii.
67.	Bridgeport, Conn	Oct., '94	24	xxxv.
68.	Florida‡	Mar., '95	25	xix.
69.	Atlanta, Ga	Oct., '95	25	xxxiii.
70.	Pittsburgh, Pa.*	Feb., '96	26	xvii.
71.	Colorado	Sept., '96	26	xxix.
72.	Chicago, Ill	Feb., '97	27	xvii.
73.	Lake Superior	July, '97	27	xxx.
74.	Atlantic City, N. J.*	Feb., '98	28	xvii.
75.	Buffalo, N. Y	Oct., '98	28	xxxvi.
76.	New York, N. Y.*	Feb., '99	29	xvii.
77.	California	Sept., '99	29	xlix.
78.	Washington, D. C.*	Feb., '00	30	xix.
79.	Canada	Aug., '00	30	xlv.
80.	Richmond, Va.*	Feb., '01	31	xix.
81.	Mexico	Nov., '01	32	cxviii.
82.	Philadelphia, Pa.§	May, '02	33	xxxv.
83.	New Haven, Conn	Oct., '02	33	xlvii.
84.	Albany, N. Y.*	Feb., '03	34	xxiii.
85.	New York, N. Y	Oct., '03	34	lxi.
86.	Atlantic City, N. J.*	Feb., '04	35	xxiii.
87.	Lake Superior	Sept., '04	35	xlii.
88.	Washington, D. C	May, '05	36	xlii.
89.	British Columbia	July, '05	36	liii.
90.	Bethlehem, Pa	Feb., '06	37	xli.
91.	London, England	July, '06	37	xlviii.
92.	New York, N. Y	April, '07	38	lii.
93.	Toronto, Canada	July, '07	38	lix.
94.	New York, N. Y	Feb., '08	39	xli.
95.	Chattanooga, Tenn	Oct., '08	39	xlviii.
96.	New Haven, Conn	Feb., '09	40	xli.
97.	Spokane, Wash	Sept., '09	40	xlviii.
98.	Pittsburgh, Pa	Mar., '10	41	xxxviii.
99.	Canal Zone	Nov., '10	41	xlv.
100.	Wilkes-Barre, Pa	June, '11	42	xxxiv.
101.	San Francisco, Cal	Oct., '11	42	xliv.
102.	New York, N. Y.*	Feb., '12	43	lxxvii.
103.	Cleveland, Ohio	Oct., '12	44	vii.
104.	New York, N. Y.*	Feb., '13	45	xv.
105.	Butte, Mont	Aug., '13	46	vii.
106.	New York, N. Y	Oct., '13	47	vii.
107.	New York, N. Y.*	Feb., '14	48	xv.
108.	Salt Lake City, Utah	Aug., '14	49	vii.
109.	Pittsburgh, Pa	Oct., '14	50	vii.
110.	New York, N. Y.*	Feb., '15	51	xvi.
111.	San Francisco, Cal	Sept., '15	52	vii.
112.	New York, N. Y.*	Feb., '16	54	xvi.
113.	Arizona	Sept., '16	55	vii.
114.	New York, N. Y.*	Feb., '17	56	vii.
115.	St. Louis, Mo	Oct., '17	57	vii.
116.	New York, N. Y.*	Feb., '18	59	xvii.
117.	Colorado	Sept., '18	60	vii.
118.	Milwaukee, Wis	Oct., '18	60	xxii.
119.	New York, N. Y.*	Feb., '19	61	xi.
120.	Chicago	Sept., '19	62	vii.
121.	New York, N. Y.*	Feb., '20	63	ix.
122.	Lake Superior District	Aug., '20	66	ix.
123.	New York, N. Y.*	Feb., '21	66	xiii.
124.	Wilkes-Barre, Pa	Sept., '21	66	xix.
125.	New York, N. Y.*	Feb., '22	67	ix.
126.	San Francisco, Cal	Sept., '22	68	xxix.
127.	New York, N. Y.*	Feb., '23	69	xxix.
128.	Canada	Aug., '23	69	xxxvi.
129.	New York, N. Y.*	Feb., '24	70	xxxi.
130.	Birmingham, Ala	Oct., '24	71	xxxiii.
131.	New York, N. Y.*	Feb., '25	71	xxxviii.
132.	Salt Lake City, Utah	Sept., '25	73	xvi.
133.	New York, N. Y.*	Feb., '26	73	xxii.
134.	Pittsburgh, Pa	Oct., '26	74	xv.
135.	New York, N. Y.*	Feb., '27	75	xvii.
136.	New York, N. Y	Feb., '28	76	104‖
137.	New York, N. Y	Feb., '29	29¶	116‖
138.	San Francisco, Cal	Oct., '29		496‖

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* Annual meeting for the election of officers. The rules were amended at the Chattanooga meeting May, 1878, changing the annual election from May to February.

† Begun in May at Easton, Pa., for the election of officers, and adjourned to Philadelphia.

‡ Begun in February at New York City, for the election of officers, and adjourned to Florida.

§ Begun in February at New York City, for the election of officers, and adjourned to Philadelphia.

‖ See MINING AND METALLURGY of year of meeting on page indicated for complete story of meeting.

¶ Many of the papers presented at the 137th general meeting are in the TRANSACTIONS for 1929 (four volumes).

Institute Meetings in 1929

The Annual Meeting was held at Institute headquarters in New York City, February 18–22, 1929; for program of meeting see MINING AND METALLURGY for February, page 67, and for complete story MINING AND METALLURGY for March, page 116.

REGIONAL AND DIVISION MEETINGS

Meeting	Place	Date	Report in MINING AND METALLURGY, Page
Joint meeting Institute of Metals and Iron and Steel Divisions, A. I. M. E., Iron and Steel Division, A. S. M. E., American Welding Society, and American Society for Steel Treating.	Cleveland	Sept. 9–13	Aug.,[1] 380 Oct.,[2] 478 Oct.,[3] 480 Nov.,[4] 540
Regional meeting	Spokane	Oct. 3–5	Aug.,[1] 383 Oct.,[1] 442 Nov., 501
Petroleum Division, A. I. M. E.	Tulsa	Oct. 3–5	Sept.,[1] 411 Oct.,[1] 442 Nov., 505
Petroleum Division, A. I. M. E.	Los Angeles	Oct. 4–6	Oct.,[1] 442 Nov., 504
138th General Meeting American Institute of Mining and Metallurgical Engineers.	San Francisco	Oct. 7–10	Aug.,[1] 384 Sept.,[1] 410 Oct.,[1] 442 Nov., 496

[1] Preliminary story of meeting containing program.
[2] Iron and Steel Division, A. I. M. E.
[3] Institute of Metals Division, A. I. M. E.
[4] Secondary Metals Symposium.

OFFICIAL INSTITUTE REPORTS FOR THE YEAR 1928

Report of the Secretary

To the Board of Directors and the Members of the American Institute of Mining and Metallurgical Engineers

Gentlemen:

The year 1928 was another in the history of the Institute which was marked by quiet growth and steady progress. The total membership as of December 31, 1928, was 8703 as compared with 8438 on December 31, in 1927. Again the main increase has been in Members and the number of Junior Associates is less. As will be seen also by the Treasurer's report, the increase is reflected in larger receipts for dues indicating that fewer have had to be dropped for non-payment. There has been a gratifying increase in the number applying for reinstatement and it seems clear that interest in and attachment to the Institute has notably increased.

The year was signalized by the merging of the Rocky Mountain Club with the Institute which brought to our roll one hundred and forty who were not previously members from among the men who have made the West. It also brought to the Institute a strong tradition of good fellowship and a fund of over $100,000, the interest from which is to be expended so far as may be, for purposes beneficial to the Western states. The Endowment Committee expects to see other and similar funds entrusted to the Institute and the Board has already appointed a special committee to consider all projects for expenditure of the revenue from them. It is felt that with the organization and personnel available in the Institute even modest revenues can be made to yield large returns in the way of scientific and technical knowledge.

Within the year also, a third great Division has been organized within the Institute, building on the foundation of the old Iron and Steel Committee. The new Division signalized its independence by joining the Lake Superior Mining Institute in a visit to the Menominee Range in September and by furnishing material for a special volume of 353 pages, "Iron and Steel Technology in 1928," printed and distributed in December. The Institute of Metals Division held its fall meeting with good attendance as one feature of National Metal Week at Philadelphia in October, and published its volume of Proceedings, running to 836 pages, in August. "Petroleum Development and Technology in 1927," 844 pages, came out in June and recorded the work of the Petroleum Division. A most interesting Fall meeting was also held by the Division at Tulsa, Okla., Oct. 18–19, and an exhibit made at the International Petroleum Exposition there the week following.

Two regional meetings were held within the year, the first at Boston in August and the second at Los Angeles in September. The Boston meeting was under the auspices of the Boston Section and the newly-formed Committee on Geophysical Methods of Prospecting. The papers attracted to this and the program of the Committee at its first meeting in February, 1928, were so numerous and valuable that the Board authorized publication of a special volume on "Geophysical Prospecting" now being distributed. Nearly 1200 members of the Institute have signified their interest in this subject, so that the Committee now represents one of the largest groups in the organization, more than in many independent societies. Western members got together at Los Angeles in September, the Southern California Section being responsible for a portion of the program of Convention Week, in which the Western Division of the American Mining Congress and the California Development Association, joined: The papers contributed by the Institute related to non-metallics and non-ferrous mining and metallurgy.

The 27 Local Sections of the Institute each held one or more meetings, a number putting on monthly programs. Every active Section was visited at some time within the year by the President, or one of the Secretaries, or by two of them together in order to effect personal contact between the staff and local members. Each Section, also, was invited to send a delegate to the Annual Meeting and with few exceptions they did so. It is hoped still further to build up these relations but necessarily more of the work must fall on other officers of the Institute, since neither time nor money are available to permit much more travel on the part of those now doing the work. In a further effort to promote contact the By-laws were amended to permit payment of certain expenses incurred by Directors in connection with meetings of the Board. So far as one year's experience goes this had made no material difference. Out of a possible attendance of 248 at 10 meetings held, 99 was scored. Of the Directors, three attended 9 meetings; two, 8; three, 7; one, 6; one, 5; three, 3; one, 2; three, 1; six attended none. At only one meeting was there a failure to secure a quorum. Only $117.09 was charged to account of Directors' travelling expense, the members, as in the past, if able to come at all, arranging their trips in connection with other business or making no charge.

The publications printed for the year included the following:

12 Issues of "Mining and Metallurgy," 570 pages.
TRANSACTIONS Volume 76, 755 pages.
Proceedings of the Institute of Metals Division, 836 pages.
Petroleum Development and Technology, 1927, 844 pages.
Iron and Steel Technology, 1928, 353 pages.

Flotation Practice, furnished by the Milling Methods Committee, 257 pages.

Technical Publications, individual pamphlets, 113 in number.

In addition, there were a number of preprints and mimeographed advance copies of papers distributed and the special volume on Geophysical Prospecting to which reference has already been made, was virtually completed though not printed until January, 1929. Taking into account the size of the edition of each paper it may be estimated that 16,000,000 printed pages of technical data were furnished the members in 1928 at a total cost of $95,906.18, publication constituting the main expense as it does the principal business of the Institute.

The Board now has under consideration, and has submitted to the members for discussion, further changes in the By-laws affecting the distribution of the publications and the position of junior members. The first is designed to meet inequalities in distribution by putting all the material that now goes into the special volumes into TRANSACTIONS, furnishing each bound in cloth without extra charge, supplying one volume of his choice free to each member, and making a moderate charge to cover costs for any additional volumes requested by any member. It is believed that this plan provides the fairest distribution of the mounting cost of publication resulting from increasing volume and the increase in printing charges. Each year a consolidated index, such as appears in TRANSACTIONS Volume 76, will be furnished to each member so that he may have no difficulty in referring to anything printed by the Institute whether in the particular volumes in his own library or not.

The change proposed in regard to membership, abolishing the grade of Junior Associate and establishing those of Junior Members and Student Associates, is designed to make it easier for young men to make contact with the Institute and to progress to full membership as they become qualified. It also places our membership more nearly on the same basis as that of the other National Engineering Societies, though it still remains true that our dues are the lowest of the group.

The relations of the Institute to various societies serving similar or like fields has remained cordial and pleasant and our inter-society business and professional relations, such as participation in United Engineering Society, Engineering Foundation, Engineering Societies Library, the Employment Service and similar organizations remain with but little change. The financing of the Library has been placed on a slightly different basis, so that temporarily the assessment against the Institute has been decreased, and a slight decrease in connection with participation in the Engineering Standards Association will also accrue this year. It remains true, however, that the fields for usefulness are so numerous and so broad that the income of the organization is inadequate and only the closest economy in detail permits the present activities to be maintained.

The Woman's Auxiliary has continued its excellent work and its members meet regularly in many parts of the country. Within the year it has been incorporated and has been the recipient of gifts amounting to a number of thousands of dollars for permanent support of its student fellowships. A fund of $10,000 is also being raised, the income to be used to supplement dues in meeting clerical and other incidental expenses. A room at headquarters has been set aside for the Auxiliary office, the Institute for the present paying the rent. Members of the Auxiliary have joined with those affiliated with other engineering societies in leasing and opening a Club House for social purposes at 37th Street and Lexington Avenue, not far from Institute headquarters in New York. This activity of the Auxiliary, of the Open Hearth Conference, and various other groups might also be cited as illustrating the capacity of the miners, metallurgists and all related to their professions, to organize and conduct movements designed to better conditions in their field and is a welcome indication that as a class we have ideals and purposes beyond merely making holes in the ground and making money out of rock.

H. Foster Bain,
Secretary.

Report of the Membership Committee

The name of this Committee, which was formerly known as the Committee on Increase of Membership, was changed in the revised Constitution adopted last year. At the same time, the form of organization and the functions of the Committee slightly altered. Previously it had concerned itself only with endeavoring to secure applications for membership from those who are, or should be, interested in the field of activities of the Institute and in a position to benefit from membership in it. The Committee was large and widely scattered, with the result that it was impracticable for it to hold meetings and discuss any of the larger problems involved in stimulating growth along sound and healthful lines. At the suggestion of last year's Chairman, the Assistant Secretary of the Institute, an executive committee of the main committee, consisting of two past presidents and three other members, each a prominent figure in one branch of the mineral industry, and with the past Chairman as executive secretary, was created to make a study of the general problems of membership. The first problem it took up was the membership structure of the Institute in its relation to beginners in the industry. After reviewing the organization of the three other national engineering societies and the experience of the Institute during the past ten years with the Junior Associate grade of membership, the Committee recommended to the Board of Directors, on Nov. 21, 1928, that this grade of membership be abolished and the two grades of Student Associate and

Junior Member be substituted therefor. The purpose of this change is to make it easier for young engineers employed in the mineral industry, who, when they were undergraduates, did not know that they would enter the industry, and consequently did not become Junior Associates, to become members in a Junior grade at reduced annual dues.

The changes in publication policy which the special committee of the Board has since recommended will make the new grade of Junior Member even more attractive than had been planned, since those in that grade will receive a cloth bound copy of one volume of the TRANSACTIONS for $1 less than it has heretofore cost a Junior Associate to receive the TRANSACTIONS. It is, therefore, expected that the new grade of membership will not only provide for a class of young men who were not being reached by the Institute, but will provide a more satisfactory relationship for all young men in the industry. Since the Institute loses by death and by resignation (due to going into other lines of work) about 300 members each year and only about the same number of young men graduate from schools of mining and metallurgy in the same period, it is evident that careful attention must be given to securing the interest of the younger men if the Institute is to continue to grow.

An important change in the functioning of the main committee has been its coordination with the activities of Local Sections, through the Sections' appointing sectional committees on membership, the Chairmen of which should be ex officio members of the Committee on Membership. A number of Local Sections have done this and it is hoped the practice may be more generally adopted in 1929. The regular activities of the Committee have continued throughout the year, and in that period 607 applications for membership were received, 601 were elected to membership and 597 accepted election, thus comparing favorably with preceding years.

E. DEGOLYER,
Chairman.

Report of the Admissions Committee

This Committee, formerly known as the Membership Committee, had its title changed as above by the new Constitution which was adopted February, 1928. During the year 1928, 607 applications for membership were submitted to the Committee for its action; and after careful consideration 601 were recommended to the Board of Directors for election to the various grades. Of this number 597 accepted election and became members during 1928. The changes in the membership resulting therefrom are set forth in tabular form on the following page.

The Rocky Mountain members, who now appear in this list for the first time, were added pursuant to the action taken at the Annual Meeting on Feb. 21, 1928. At that time it was voted to offer members of the Rocky Mountain Club, not already members of the Institute at the time of the merging of the two organizations, a special limited life membership. They constitute a permanent addition to our membership without burden on the Institute's revenues.

Total membership, Jan. 1, 1928		8438
Accepted election, 1928:		
Members	227	
Associates	100	
Junior Associates	205	
	532	
		532
Reinstatements		65
Rocky Mountain Members		140
Total		9175
Losses:		
Loss by deaths, 1928	85	
Loss by resignations, 1928	149	
Suspended for nonpayment of dues	305	
Junior Associates failed to change status	54	
	593	593
Total membership, Jan. 1, 1929		8582

To this Committee it does not seem that satisfactory progress was made in the net membership increase during 1928, which ended the year with a total of 8582 as against a membership of 8438 on Dec. 31, 1927, an increase of but 144 which, including as it does the 140 members coming to us from the Rocky Mountain Club, really represents for 1928 a net increase of but 4 members.

It has ever been conceded that even vigorous efforts by the Institute to enlarge its membership by eligible persons is not only vital to its future but that it is an entirely dignified proceeding. It would seem that large increases could be had by consistent continuous work on some well laid plan evolved by those in charge. The functions of this Admissions Committee are merely to pass on the qualifications of applicants and recommend to the Board of Directors the election of those who have measured up.

George D. Barron,
Chairman.

Report of Committee on Papers and Publications

The following is a report of the papers submitted to the Papers and Publications Committee during 1928:

The Committee held nine regular meetings during the year 1928. During that period there were presented to the Institute 214 papers which were acted upon as follows:

Accepted as *Technical Publications*	85
Declined for publication	16
Action pending, Dec. 31	25
Manuscripts held for annual volumes, Dec. 31	7
Withdrawn temporarily	1
Accepted as Preprints	20
Printed for first time in volumes	35
Published in MINING AND METALLURGY	12
Papers in *Technical Publication* No. 156 (Coal Classification) and not accounted for elsewhere in this table	10
Mimeographed only*	3
Total	214

* A considerable number of papers were mimeographed for the annual meeting.

Of the 85 papers accepted as *Technical Publications* 8 were reprinted in the TRANSACTIONS, 7 in *Proceedings* of the Institute of Metals Division (1928), 7 in Iron and Steel Technology in 1928, 13 in Geophysical Prospecting, and 50 were in *Technical Publication* form on Dec. 31. Of the 20 papers accepted as Preprints 1 was reprinted in the TRANSACTIONS, 13 in Petroleum Development and Technology in 1927, 2 in Geophysical Prospecting, and 4 were in Preprint form on Dec. 31. Of the 35 papers printed for the first time in the various volumes 8 were included in the TRANSACTIONS, 2 in *Proceedings* of the Institute of Metals Division (1928), 1 in Iron and Steel Technology in 1928, and 24 in Petroleum Development and Technology in 1927. Of the 12 papers accepted for MINING AND METALLURGY 1 was accepted for Geophysical Prospecting and 1 for the TRANSACTIONS.

There were published in 1928 113 *Technical Publications* containing a total of 1928 pages, as against 88 in 1927, totaling 1528 pages. The *Technical Publications* published in 1928 were classified as follows: Group A, Metal Mining, 17 papers (361 pages); B, Milling and Concentration, 4 papers (39 pages); C, Iron and Steel, 15 papers (256 pages); D, Non-ferrous Metallurgy, 13 papers (191 pages); E, Institute of Metals, 36 papers (588 pages); F, Coal and Coke, 15 papers (392 pages); G, Petroleum and Gas, 10 papers (246 pages); H, Non-metallic Minerals, 3 papers (33 pages); I, Mining Geology, 18 papers (356 pages); K, Mining Administration, etc., 3 papers (47 pages); L, Geophysical Prospecting, 21 papers (430 pages).

Five volumes were published by the Institute in 1928, the contents of which were selected by the Committee. The contents appeared for the first time as shown in the following table:

	Published First in 1927, Pages	Published as 1928 *Technical Publications*, Pages	Published First in Volume, Pages
Petroleum Development and Technology in 1927	36		808
Proceedings of the Institute of Metals Division, 1928	284	340	212
Iron and Steel Technology in 1928	32	235	96
Flotation Practice	110		147
TRANSACTIONS, Vol. 76*	79	343	246

* Eighty-seven pages of Vol. 76 of the TRANSACTIONS not accounted for in the above table were reprinted from other volumes.

As provided in Article VIII, Section 4 of the By-laws, the Papers and Publications Committee formulated rules of procedure setting forth its policies regarding the acceptance or rejection of papers submitted for publication; these were approved by the Board of Directors on March 30, 1928, and a copy of the rules of procedure may be obtained by application to the Secretary of the Institute.

Within the year the Membership Committee recommended material changes in the classification and rights of members and this in turn led to recommendations by a special committee for extensive changes in the plan of publication of the TRANSACTIONS and special volumes. These recommendations were approved by the Board and Amendments to the By-laws necessary to put them into force will come before it for vote at the February meeting. In brief, the change contemplates that hereafter there shall be printed and distributed to all members a YEAR-BOOK including the present list of members, committees and business reports and a comprehensive consolidated index of all the publications of the Institute for the year concerned. The technical papers now included in the TRANSACTIONS, as well as those now issued in the special volumes, shall all go into a series of TRANSACTION volumes suitably allocated to the work of the separate Divisions and of the Committees. One such volume each year, as chosen by him, shall be furnished free in cloth binding to each member and he may buy any or all of the additional volumes similarly bound, at a price estimated to cover the average cost. Under this plan each member will receive annually a comprehensive index of all the publications, a bound volume of the papers in his special field if they are in number sufficient to make up a volume, and will be permitted to buy substantially at cost the volumes covering the fields in which he is less interested. MINING AND METALLURGY will continue to go to all members and *Technical Publications* will be printed and distributed as heretofore. The Papers and Publications Committee will, as always, be finally responsible for inclusion of material in the TRANSACTIONS.

H. G. MOULTON,
Chairman.

Report of the Library Committee

The Library belonging to the Institute is now administered as part of that of the United Engineering Society, the members of the Library Committee representing the Institute on the Library Board of the United Engineering Society. The main collection now consists of 115,900 books, 3680 pamphlets, 5811 maps, and 4402 searches, making a total of 129,793, a net gain of 7701 within the year. There is in addition a lending collection of 1004 volumes and one of duplicates amounting to 11,310. The library grows steadily both by gift and purchase. The cataloging is now substantially current covering 124,100 items represented by 362,145 cards. There are many rare and notable books, especially those relating to early engineering. It is truly a great treasure house of knowledge. While no severe effort is made to record all visitors the books showed 23,470 to have used the Library in 1928. In addition 5102 telephone requests for information were answered, 202 searches and 196 translations made and 34,964 photo prints furnished. The service department is maintained by small fees which cover the cost. The total amount appropriated for operating expense for the year was $46,300, of which $8,240.02 was furnished by the Institute, $5,399.45 being derived as income from the Douglas Library Fund and the remainder from current receipts. Within the year a slight change was made in the method of meeting the expenses of the Library, a flat charge of $5,000 per year being made against each of the four Founder Societies and this being supplemented by a surcharge based upon membership. The effect of this has been for the present to decrease slightly the amount that the Institute would otherwise need to pay.

Placing in one collection all the books of the four societies greatly increases the number of books accessible to the members of each and opening the Library to the public is an important factor in keeping the tax exemption privilege enjoyed for the building. The Director and Staff of the Library make every effort not only to accommodate those who come in person to consult the collection but members resident elsewhere who communicate by letter or wire. The facilities of the Library are truly available to all.

GEORGE C. STONE,
Chairman of Library Committee.

Report of Committee on Mining and Metallurgy

MINING AND METALLURGY closed the year 1928 showing a small profit, the receipts, exclusive of the charge against members, dues for subscriptions, having been $43,067.81, and expenditures $40,925.58. In accord with the policy of the Board the budget is made each year with the purpose of balancing the accounts rather than the maximum of profit from the undertaking since it is the service the journal performs for members that justifies its existence. This service is being constantly increased. Within the year a number of changes have been made in the dress of the magazine, a new cover adopted, color more frequently used, and illustrations more generally employed. These changes seem to have been appreciated both by readers and advertisers.

More space has been devoted to papers read before local sections and to the news of their activities. It is believed that this has helped materially in sustaining and increasing the interest in such meetings. The general, regional, and division meetings have also been fully reported and news of Board meetings, of the work of the various committees, of the employment service and other activities of the Institute has been spread before its members month by month. The 570 pages of reading matter printed in the course of the year constitute not only a record of what the Institute was doing but of what the most representative sections possible of the members of the mining and metallurgical professions were thinking. With 9000 members scattered around the world, occupying key positions in the industry, the Institute affords through its own journal an unsurpassed medium for broad contact and enables the readers with the minimum of expenditure of time and at no extra money expense to keep a general contact with the whole range of subjects touched by miners and metallurgists. In these days of intense specialization but rapid change such contact is particularly valuable.

The more formal papers presented to the Institute are published in the TRANSACTIONS, *Technical Publications* and the special volumes. MINING AND METALLURGY carries abstracts of all papers published by the Institute and prints in full the shorter, less detailed papers and those which reflect work in progress rather than finished. Such papers have a news value which is supplemented by editorial comment, discussion by members and others, and the actual news of the organization itself. Without therefore attempting to fill the place of a general news sheet, it has a timeliness that gives it an immediate value much appreciated by the readers.

In accord with the general policy of the Board every effort will be made constantly to improve the quality of the magazine to the full extent that its revenue will permit.

Your Committee feels that, through the intelligent cooperation of the official staff in the office, MINING AND METALLURGY has been improved in general appearance, in subject matter and in its financial position. We regard it as one of the strong arms of the Institute. Even if it should cost more than its income it renders such a valuable service, through publicity, that its publication is fully justified. All efforts, however, have been directed toward an equalization of expenditures with income.

W. L. Saunders,
Chairman.

Treasurer's Annual Report, Year of 1928

Statement of Receipts

For the year ended December 31, 1928

Dues:		
Arrears	$ 4,878.18	
Current	96,174.46	
New Members	6,281.50	
Dues for future periods received in advance	1,745.28	$109,079.42
Initiation Fees		7,879.00
Magazine Receipts:		
Advertising Gross Receipts	$39,247.77	
Magazine Sales	3,820.04	43,067.81
Sale of Transactions Volumes 75 and 76	$ 2,525.93	
Sale of Transactions and Special Editions	7,586.00	
Sale of Year Book	141.60	
Sale of Authors' Reprints	2,032.97	
Sale of Pamphlets	1,346.29	
Sale of Bindings	4,551.90	18,184.69
Interest Received:		
On Bank Balance and Temporary Investments	$ 2,780.75	
On Life Membership Fund	2,106.43	
On James Douglas Library Fund	5,399.45	
On General Reserve Fund	1,995.33	12,281.96
Miscellaneous Receipts		131.16
Total Receipts		$190,624.04

STATEMENT OF EXPENDITURES

For the year ended December 31, 1928

Magazine Expenditures			$ 40,925.58
All Other Publication Expenses:			
Transactions:			
Volumes 1 to 75 inclusive	$ 989.45		
Volume 76	13,288.83		
Volume 77	951.80	$15,230.08	
Pamphlets		13,999.90	
Petroleum Division—Volume '28		6,424.78	
Petroleum Division—Volume '29		383.47	
Institute of Metals Division—Volume '28		5,486.97	
Institute of Metals Division—Volume '29		805.55	
Iron and Steel Volume		3,218.37	
Milling and Concentration Volume		1,815.26	
Geophysical Volume		2,470.95	
Authors' Reprints		1,338.48	
Year Book		3,460.14	
Special Editions		346.65	54,980.60
Joint Activities:			
Library Assessment		$ 8,240.02	
Employment Service		437.61	
American Engineering Standards Committee		1,500.00	10,177.63
Meetings—Technical Committee, Local Sections:			
Local Section Appropriations		$ 4,375.00	
Meetings		10,231.49	
Local Section Traveling Expense		2,466.41	
Technical Committee and Divisions		1,226.19	18,299.09
Accounting, Membership and Membership Increase Departments:			
Accounting Department Salaries and Expense		$ 8,589.11	
Membership Department Salaries and Expense		2,990.03	
Increase of Membership Department Salaries and Expense		4,580.82	16,159.96
General Office Salaries			36,295.48
Miscellaneous General Office Expense:			
Office Space		$ 1,944.00	
Stationery and Supplies		1,468.21	
Telephone and Telegraph		1,693.22	
Postage (General Office Mail)		279.04	
Furniture and Fixtures (Including $1,500.00 Depreciation)		2,356.98	
Miscellaneous Office Expense		542.81	
Miscellaneous Institute Expense		2,004.39	
Traveling Expenses of President and Secretary		2,501.10	
Traveling Expenses of Directors		117.09	
Legal Expenses		250.00	
Insurance		144.54	13,301.38
Total Expenditures			$190,139.72

ASSETS

Interest in Real Estate and other Assets of United Engineering Society except Trust Funds		$491,642.36	
Books in Library		40,000.00	
Investments			
Life Membership Fund	$ 59,278.45		
James Douglas Library Fund	99,138.50		
Robert W. Hunt Fund	7,502.25		
J. E. Johnson, Jr. Fund	2,922.63		
James Douglas Medal Fund	3,270.00		
William Lawrence Saunders Mining Medal Fund	7,000.00		
Anthony F. Lucas Fund	3,115.00		
Rocky Mountain Club Fund	109,500.00		
General Reserve Fund	35,733.12		
Total of Funds Invested		327,459.95	
Cash	$ 9,617.07		
Cash Invested Temporarily	9,718.50		
Accounts Receivable	751.84	20,087.41	
Furniture and Fixtures	$ 13,214.57		
Less Reserve for Depreciation	1,500.00	11,714.57	
Prepaid Expenses		514.60	
Inventory Paper on hand	$ 2,572.71		
Transactions and Special Editions	19,430.00		
Technical Publications Binders	244.24		
Postage on hand	871.25		
Pins and Fobs	243.99		
Supplies on hand	241.57	23,603.76	$915,022.65

LIABILITIES

Accounts Payable		$ 8,664.27	
Due the Mining Methods Committee		653.38	
Due the Life Membership Fund	$ 59,283.92		
James Douglas Library Fund	100,000.00		
Robert W. Hunt Fund	8,448.01		
J. E. Johnson, Jr. Fund	3,197.10		
James Douglas Medal Fund	3,145.27		
William Lawrence Saunders Mining Medal Fund	7,155.87		
Anthony F. Lucas Fund	4,448.58		
Rocky Mountain Club Fund	110,097.02		
General Reserve Fund	36,656.35		
Total due the Funds		332,432.12	
Reserve for Life Membership Fund		40,000.00	
Surplus as follows:			
Contributions by Andrew Carnegie and others to Real Estate	$491,642.36		
Inventories	35,318.33		
Available Surplus	6,312.19	533,272.88	$915,022.65

We hereby certify that we have audited the books and account of the American Institute of Mining and Metallurgical Engineers and that the above statement of Assets and Liabilities is in accord therewith as at December 31, 1928.

KARL EILERS,
Treasurer.

January 16, 1929.
Loomis, Suffern & Fernald,
Certified Public Accountants.

Necrology

The following is a list of members who died in 1928. It is compiled from reports to the Secretary's office. Biographical sketches published in MINING AND METALLURGY are indicated in the last two columns.

YEAR OF ELECTION	NAME	DATE OF DEATH	ISSUE CONTAINING BIOGRAPHY 1928	PAGE
1914	ADAMS, DAVID T.	July 22	October	471
1905	AGUILERA, PEDRO	May 3	October	469
1902	ALTHOUSE, H. W.	Nov. 3	December	559
1920	AVERILL, C. C.	July 9	October	471
1915	BALDWIN, C. KEMBLE	Aug. 9	October	471
1926	BARQUIST, WALTER C.	Nov. 14	June†	298
	BARRON,* CLARENCE WALKER	Oct. 2	November	511
1917	BEASLEY, FRED E.	Aug. 31	October	471
1901	BELDEN, S. B.	May 15	July	333
1927	BINNEY, EDWIN, JR.	Dec. 29	February†	96
1915	BONSALL, JAMES MALCOLM	Feb. 12	June	284
1917	BOYLE, JOHN	July 12	December	561
1913	BRADEN, EUGENE B.	Jan. 29	March	157
1889	BULKLEY, FRANK	Mar. 25	May	240
1896	CALLAGHAN, J. T., JR.	Apr. 1		
1892	CHALMERS, GEORGE	Feb. 4	April	197
1898	CLARK, JOSIAH H.	May 7	July	333
1887	COPELAND, F. K.	Nov. 10	December	561
1915	CUNNINGHAM, B. L.	Dec. 6	August†	294
1888	DANA, R. T.	Aug. 20	December	559
1907	DENNIS, CLIFFORD G.	Dec. 28	March†	168
1920	DEVENNY, THOMAS	Feb. 13	April	197
1882	D'INVILLIERS, EDWARD V.	Jan. 4	March	157
1884	DUNCAN, M. M.	Dec. 17	February†	96
1924	DYRSSEN, WALDEMAR	Aug. 4	October	471
1906	EDMONDSON, H. W.	Dec. 21	February†	95
1884	EMMERTON, F. A.	Oct. 9	June†	298
1911	EU, S. H.	July 20	February†	95
1917	FARRAND, IRA W.	Sept. 6	October	469
1900	FELL, E. NELSON	Mar. 25	May	231
1918	FENTON, CLARENCE M.	October	June†	298
1908	FERRARIS, ERMINIO	Sept. 22	June†	298
1914	GALLOWAY, A. D. R.	December	April†	215
1924	GRANT, A. W., JR.	Apr. 5	June	285
1917	GRANT, CLIFFORD G.	Jan. 26	January†	37
1918	GRANT, FRANK E.	Jan. 14	September	417
1921	GUNDEROTH, C. J.	Feb. 15		
1897	HARMS, ERNEST	Aug. 12	November	512
1891	HARTRICK, J. S.	Feb. 9	June	285
1925	HOCHSCHILD, BERTHOLD	Jan. 24	April	197
1917	HUTTON, L. W.	Nov. 3	January†	37
1894	JENNINGS, S. J.	Nov. 17	December	560
1925	JONES, J. RAVENSCROFT	Jan. 29	March	157
1880	JONES, J. T.	May 4	June	284

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* A Rocky Mountain Club member of the Institute.

† 1929 Issue.

Year of Election	Name	Date of Death	Issue Containing Biography 1928	Page
1901	L'Engle, E. Fleming	Apr. 21	June	285
1917	Lobner, Charles	Oct. 19	February†	95
1904	Longworth, William	Dec. 5		
1913	Loughridge, Charles	Apr. 27	August	376
1913	Ludlum, Albert C.	Feb. 16	March	158
1914	Marshall, George B.	1928	January†	37
1924	Maxwell, C. B.	Sept. 14	February†	95
1901	McChrystal, Jackson C.	July 10	September	417
1924	McCormack, Clyde Preston	July 8		
1891	Nason, F. L.	Sept. 12	October	472
1923	Needham, James	Jan. 31	July	332
1895	Newson, John F.	Oct. 24	April†	215
1898	Nichols, W. H., Jr.	May 26	May†	257
1885	Norris, R. V.	Apr. 20	May	231
1922	O'Toole, William J.	August	July†	349
1920	Payne, Edgar D.	Jan. 22	June	285
1871	Pechin,* Edmund C.	Feb. 6	March	158
1886	Pierce, J. B.		February†	95
1881	Pluemer, A.	August	April†	213
1889	Polledo, Y.	May 7		
1898	Ralston, John C.	July 14	September	418
1895	Raymond, R. M.	Aug. 23	October	470
1882	Riordan, Denis M.	July 12	November	511
1924	Risacher, Felix E.	June 14	April†	213
1903	Roberts, Robert P.	Sept. 11	November	510
1916	Roberts, T. C.	Sept. 28	November	510
1918	Robertson, Norman A.	May 24	July	333
1912	Rood, Vernon S.	May 1	June	284
1925	Russell, Emerson C.	Nov. 14	May†	257
1926	Schlegel, C. D.	Nov. 26	May†	257
1891	Schroedter, Emil	Oct. 31	February†	95
1917	Segall, Julius	Nov. 27	April†	214
1928	Sharpley, Harold	Nov. 12	April†	214
1907	Simon, T. B.	Oct. 28	December	561
1922	Smith, Frank Morse	Apr. 1	June	285
1915	Smith, Howard C.	May 30	July	333
1924	Sullivan, M. H.	Feb. 12	March	157
1927	Sullivan, R. D.	Feb. 26	April	197
1927	Sullivan, Ralph D.	Feb. 26	April	197
1905	Taylor, E. J.	Dec. 24	January†	37
1887	Taylor, William H.	Feb. 29	May	239
1898	Tays, E. A. H.	July 22	November	512
1920	Thomas, Walter H.	Dec. 23	April†	213
1925	Walker, E. V.	Nov. 19	January†	37
1905	Walsh, Edward J.	May 30	July†	349
1903	Ward, William F.	Sept. 20	May†	257
1871	Ward, Willard P.	Jan. 17	February	88
1921	Weston, E. M.	January	April	197
1926	Whitworth, E. A.	June 1	November	512
1927	Zern, E. N.	Apr. 8	May	239

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* A founder of the Institute.

† 1929 Issue.

CERTIFICATE OF INCORPORATION

[Amended Nov. 22, 1912, and June 27, 1919.]

We the undersigned, being all persons of full age and citizens of the United States, and a majority residents of the State of New York, desiring to form a corporation pursuant to the provisions of the Membership Corporations Law for the purpose of incorporating, as provided in Section 5 of Article I. of said law, the existing unincorporated association known as American Institute of Mining Engineers, do hereby make, acknowledge and file this Certificate for that purpose, and do Certify as follows:

I. That the American Institute of Mining Engineers is an unincorporated association organized and existing with the object of promoting the arts and sciences connected with the economic production of the useful minerals and metals and the welfare of those employed in these industries by means of meetings for social intercourse and the reading and discussion of professional papers, and to circulate by means of publications among its members and associates the information thus obtained.

II. That the persons duly appointed or designated to manage the affairs of said association are designated by the rules thereof Members of its Council; that the undersigned are all members of said Council as the same was constituted on the 29th day of December, 1904.

III. That on said last-mentioned date a regularly called meeting of said association was held at its office in the Borough of Manhattan, City of New York; that thirty days before such meeting notice of the intention to incorporate said association was given by mail to each member thereof whose residence or post-office address is known; that at said meeting the following resolutions were offered, seconded and duly adopted by the unanimous vote of all its members then present, to wit:

"*Resolved*, That it is the sense of the members and associates of the American Institute of Mining Engineers in general meeting assembled that it is desirable and necessary for the well being of said association and its members and for the furtherance of the objects for which the same has been formed, that said association incorporate under the Membership Corporations Law of the State of New York;

"*And Further Resolved*, That the Members of the Council of this Association, or a majority thereof, be and they hereby are authorized, in accordance with the provisions of Section 5 of Article I. of the Membership Corporations Law, to incorporate this association for the same purposes for which it has been organized and conducted, in the manner provided in Article II. of said law;

"*And Further Resolved*, That the name of said corporation as hereby adopted by this meeting shall be American Institute of Mining Engineers;

"*And Further Resolved*, That the said incorporators shall be named in the Certificate of Incorporation as directors of such corporation until its first annual meeting, and that such directors and their successors in office shall be and they hereby are authorized to enact and adopt a Constitution and By-Laws for the government of said corporation."

IV. That the name of the proposed corporation is American Institute of Mining Engineers.

V. That the purposes for which this corporation is to be formed are: To promote the arts and sciences connected with the economic production of the useful minerals and metals, and the welfare of those employed in these industries by all lawful means; to hold meetings for social intercourse and the reading and discussion of professional papers, and to circulate by means of publications among its members the information thus obtained, and to establish and maintain a place for meeting of its members, and a hall for the reading of papers and delivery of addresses, and a library of books relating to subjects cognate to the sciences and arts of mining and metallurgy.

VI. That the territory in which the operations of this corporation are to be principally conducted is the United States of America, and the principal office for the transaction of its affairs is to be located in the City, County and State of New York.

VII. That the duration of this corporation is to be perpetual.

VIII. That the number of Directors of this corporation is to be twenty-four, consisting of one President, two Past Presidents, six Vice-Presidents, and fifteen other Directors; that at the annual meeting to be held in February, 1913, there shall be elected to serve, jointly with the six members of the former Board whose terms will not then have expired, one President, who shall serve as such for one year, and as Past President for two years; two Past Presidents to be selected from former Presidents of the Institute, one to serve for one year and one to serve for two years; six Vice-Presidents, of whom two shall be elected to serve for one year, two for two years and two for three years, and nine Directors, of whom two shall be elected to serve for one year, two for two years and five for three years.

After the year 1913 there shall be elected at each annual meeting one President to serve as such for one year and as Past President for two years; two Vice-Presidents to serve for three years and five Directors to serve for three years.

At each subsequent annual meeting so many Directors only shall be elected as shall be necessary to fill vacancies then existing in the Board, and each Director then chosen shall be elected to serve for three years unless the vacancy which he shall be chosen to fill has been caused otherwise than by expiration of the former incumbent's term of office, in which event he shall be elected to serve for the balance of his predecessor's unexpired term.

IX. That the time for holding the annual meeting of this corporation shall be the third Tuesday of February in each year.

X. That the names and post-office addresses of the subscribers hereto who are to be directors of this corporation until first annual meeting are

NAMES	POST-OFFICE ADDRESSES
James Gayley	71 Broadway, New York City;
Frank Lyman	88 Wall Street, New York City;
James F. Kemp	Columbia University, New York City;
Charles H. Snow	New York University, New York City;
Frank Klepetko	24 State Street, New York City;
Thomas A. Rickard	261 Broadway, New York City;
James Douglas	99 Wall Street, New York City;
Albert R. Ledoux	99 John Street, New York City;
Rossiter W. Raymond	99 John Street, New York City;

The above certificate, dated December 30, 1904, was signed and acknowledged by the subscribers before William A. Lockwood, Notary Public (No. 74), New York County.

I, the undersigned, Justice of the Supreme Court of the State of New York, do hereby approve of the above Certificate of Incorporation of the American Institute of Mining Engineers, and do authorize the same to be filed. SAMUEL GREENBAUM,

January 7, 1905 *Justice of the Supreme Court.*

U. OF I.
LIBRARY

CONSTITUTION

[As Revised February 21, 1928]

ARTICLE I

Name and Object

Sec. 1. This Institute is incorporated under the Membership Corporations law of the State of New York; its corporate name is American Institute of Mining and Metallurgical Engineers, Inc.; and its abbreviated title A. I. M. E.

Sec. 2. The objects of this Institute, as set forth in its certificate of incorporation, are: "To promote the arts and sciences connected with the economic production of the useful minerals and metals, and the welfare of those employed in these industries by all lawful means; to hold meetings for social intercourse and the reading and discussion of professional papers, and to circulate by means of publications among its members the information thus obtained, and to establish and maintain a place for meeting of its members, and a hall for the reading of papers and delivery of addresses, and a library of books relating to subjects cognate to the sciences and arts of mining and metallurgy."

ARTICLE II

Membership

Sec. 1. The membership of this Corporation shall include all those who on the third Monday in February, 1928, were members and all others subsequently elected and qualified in accord with the By-laws of the Institute at the time of such election.

Sec. 2. The membership shall be divided into such classes as may be from time to time fixed by the By-laws.

ARTICLE III

Directors

Sec. 1. The business, financial, professional, technical, scientific and social interests of the Institute shall be managed by a Board of Directors, elected annually by letter ballot in the manner specified in the By-laws, to serve for the terms as set forth in the Certificate of Incorporation.

ARTICLE IV

Amendments

Sec. 1. Proposals to amend this Constitution may be made by resolution of the Directors or by written petition signed by at least ten members. Such proposals must reach the Secretary not later than October fifteenth. The Board of Directors shall consider them and, in case of petitions, the proposers shall be notified of the opinion of the Board in regard to them, when they may either withdraw their proposals, or accept any changes suggested, or insist on the original form. All proposals when in final form shall be mailed to all members on or before January first, with the election ballots, and with a ballot in proper form for voting on the amendment in its original or amended form. Ballots shall be received by the Secretary until the second Tuesday in February. The Tellers of the election shall count these ballots, and report the result to the chairman of the annual meeting. Any members present in person at the annual meeting who have not voted on the amendments by ballot shall have the privilege of having their votes recorded before the result of the voting is announced. Any proposed amendment which shall have been accepted by a majority of the votes thus cast shall be adopted.

BY-LAWS

[As Revised February 21, 1928 and Amended February 19, 1929]

ARTICLE I

Members—Qualifications and Election

Sec. 1. The membership of the Institute shall comprise six classes, namely: 1. Members; 2. Honorary Members; 3. Senior Members; 4. Associates; 5. Junior Members; 6, Rocky Mountain Members. All shall be equally entitled to the privileges of membership, excepting that no member of any class whose residence shall be outside of the United States of America, except those residing in Mexico and Canada, shall be entitled to vote.

Sec. 2. *Members:* A person to be eligible for election or transfer into the class of Member must be at least 27 years of age and must have had at least six years' employment in the practice of engineering, mining, geology, metallurgy, or chemistry, during at least three years of which he must have held positions of responsibility in one or more of these fields. This shall not apply to Rocky Mountain members.

Sec. 3. *Honorary Members:* Each candidate for Honorary Membership must be recommended by at least ten Members and must be elected by ballot at a meeting of the Board of Directors by the unanimous vote of the Directors present; provided, however, that the number of Honorary Members shall at no time exceed twenty. Such election shall automatically act as a transfer of such member from any other class of membership to which he may belong.

Sec. 4. *Senior Members:* The Directors may in their discretion place on the Honor Roll of Senior Members and transfer from any other class of membership other than Honorary Membership the name of any Member or Associate who has reached the age of seventy years and who has paid dues for thirty years, and for good reasons in special cases may vary the age or number of years.

Sec. 5. *Junior Members:* A person to be eligible for election to the class of Junior Member must be qualified through education or experience to hold a subordinate position in engineering, mining, metallurgical, geological or chemical work. He shall be not more than twenty-seven years of age at the time of election. The rights of the present Junior Associates shall continue until the same expire by limitation of time.

Sec. 6. *Associates:* A person eligible for election or transfer into the class of Associate shall be one, who, in the opinion of the Committee on Admissions and the Board of Directors, is suitable for such election or transfer by reason of his interest in or connection with mining, geology, metallurgy, or chemistry.

Sec. 7. *Rocky Mountain Members:* Membership in a special class of life memberships without annual or other dues and without right to receive the printed literature distributed to other members or the right and privilege of wearing the official badge of technical members, is given to all members of the Rocky Mountain Club at the time of consolidation. These special Rocky Mountain memberships shall be claimable by any member of the Rocky Mountain Club, not already a member of the Institute, availing himself of the privilege within ninety days after mailing of notice to him, by the Secretary of the Institute.

SEC. 8. *Student Associates:* There shall be a separate class known as "Student Associates" who shall not be members and who shall have none of the privileges of members except as provided in this section. A student in good standing in an approved school, who has not taken his degree and who has been nominated by at least three instructors of the nominee (one of whom must be an Institute member), may affiliate with the Institute as a Student Associate by payment of two dollars annual fee, for which he shall receive MINING AND METALLURGY or Technical Publications and be privileged to attend meetings of the Institute and to use the Engineering Societies Library and the Employment Service. A Student Associate may remain such not longer than until December 31 of the year in which he leaves school (and in any event not more than four years), at the end of which his affiliation as Student Associate shall cease.

SEC. 9. Every candidate for election as a Member, Associate, or Junior Associate must be proposed for election by at least three Members or Associates, must be approved by the Committee on Admissions, and must be elected by the Board of Directors, and not less than three-fourths of the votes cast shall be necessary to election. At its discretion the Board may accept membership in recognized technical societies in lieu of two of the proposals. The Committee on Admissions shall formulate a set of rules of procedure in acting on applications which, after approval by the Board, shall have the force of By-laws until revoked or amended.

SEC. 10. Newly-elected candidates shall be immediately notified of their election and also of their obligations by the Secretary of the Institute. Every person so elected shall become a Member, Junior Member or Associate of the Institute in the class to which he is chosen only upon payment of his initiation fee and first year's dues, as required in Article II.

SEC. 11. If any person elected a Member, Junior Member or Associate does not within four months after the mailing of notice of his election accept the same and pay his initiation fee and dues for the year in which he was elected, his election shall be null and void and he shall be notified thereof by the Secretary of the Institute.

SEC. 12. The Board may at any time, for good and sufficient reasons, change the classification of any member.

ARTICLE II

INITIATION FEE AND DUES

SEC. 1. Each newly-elected Member or Associate shall pay, immediately on notification of his election, an initiation fee of twenty dollars; Junior Members shall pay ten dollars initiation fee, and on transfer to Associateship or Membership shall be liable for ten dollars additional. Honorary Members shall not be liable for initiation fees.

SEC. 2. The dues of Members, until otherwise determined by the Institute, shall be fifteen dollars per annum, but dues shall not be required of either Honorary or Senior Members. Dues of Associates shall be fifteen dollars per annum, and those of Junior Members ten dollars per annum for the first six years of membership as a Junior Member, and thereafter fifteen dollars. Student Associates who apply for Junior Membership within six months after leaving school shall have their first year's dues remitted. Members, Junior Members or Associates elected on or after July first of any year shall pay, on receipt of Treasurer's statement, one-half only of said amount for the year of election, and if elected in the month of October, November, or December, the candidate may defer acceptance to the first of the following year without loss of status. All dues shall be payable in advance on the first day of each calendar year except as provided in this section and notice to this effect shall be published in the last number of the monthly magazine of the Institute in each calendar year. A bill shall be mailed to each Member, Junior Member and Associate on or before the first day of the month of January in each year, stating the amount of annual dues and date when payable and the penalty and conditions incident to default in payment.

Sec. 3. Any Member or Associate may become, by the payment of two hundred and twenty-five dollars at one time, a Life Member or Life Associate, and shall not be liable thereafter to pay annual dues. The money thus received shall be invested and only the income thereof used for current expenses of the Institute.

ARTICLE III

Resignations, Suspensions and Expulsions

Sec. 1. All resignations of membership shall be made to the Secretary in writing. The resignation of a Member, Junior Member or Associate shall be accepted by the Board, subject to the payment of all outstanding dues and obligations of the resigning member. A member who shall have resigned or whose membership shall have ceased in accordance with the provisions of this Article, or who shall have been expelled, shall be obligated for no further dues beyond the then current year.

Sec. 2. Any Member, Junior Member or Associate who fails to pay his dues or other indebtedness within four months after the same become due, may at the end of such time be suspended by the Board of Directors and shall then no longer be in good standing. He shall not receive the publications of the Institute until the account is paid. If any indebtedness remains unpaid for two years after the same becomes due his name shall automatically be dropped from the rolls, unless the time of payment shall be extended by the Board. The Board may reinstate a member who has been dropped or suspended under this Section.

Sec. 3. Any member belonging to any class of membership in the Institute who shall be convicted of a crime involving, in the opinion of the Board of Directors, moral turpitude shall, upon the passage of the Board of Directors of a resolution declaring the crime for which he has been convicted to be of such character, be thereupon dropped from membership in this Institute.

Sec. 4. Any member belonging to any class of membership in the Institute may be suspended or expelled by the Board of Directors for professional misconduct or, in case of citizens of the United States of America, for disloyalty to the Government of the United States of America, after charges setting forth such misconduct or disloyalty shall have been prepared and filed in writing with the Board. The Board shall appoint a committee on members' conduct, to consist of three past-presidents of the Institute, whose duty it shall be to examine into any charges or complaints against members and, if in the judgment of the committee the facts warrant, shall prepare and file with the Board formal charges against such member as specified above. As soon as may be after the receipt of such charges the Board shall fix a date for a hearing thereupon, and shall give to the accused member notice thereof in writing, mailed to him by registered mail at his last known post office address not less than thirty days before said date, accompanied by a copy of the charges and a copy of this article.

Sec. 5. On the day fixed for the hearing, the accused member may appear before the Board, either in person or by an accredited representative; hear any witnesses who may be called in support of the charges and, at his option, cross-examine the same, and hear read any documentary evidence offered in support of the charges. The accused may, in his discretion, produce and examine witnesses in his defense, and submit documentary evidence, including a statement from himself in writing. After the conclusion of the hearing, the Board of Directors shall consider and vote to approve or disapprove the charge. If the Board shall, by a vote of two-thirds of its members, declare the charges sustained, it may suspend the member for a stated period or expel him.

Sec. 6. If the accused member shall not appear at the hearing, and shall within three months thereafter file with the Board an affidavit stating that he had not received notice of the charges against him in time to enable him to present his defense, the Board shall fix a date for a hearing within three months from the receipt of such affidavit, and shall immediately notify the accused member by registered mail of such date. Upon the rehearing, the accused shall have the same privilege of presenting his defense as he would have had on the original hearing; and after the defense is presented, the Board shall take a new vote upon the charges, the result of which shall be conclusive.

Sec. 7. If the accused member upon receipt of charges shall fail to appear after thirty days' notice as above, or shall fail to file an affidavit as set forth in Section 6 above, or after receipt of such affidavit shall then fail to appear, the Board may then by a two-thirds vote suspend or expel such member.

ARTICLE IV

Meetings of the Institute

Sec. 1. The annual meeting of the Institute for the election of Directors and transaction of other business shall be held at the headquarters of the Institute in the City of New York on the third Tuesday in February in each year, or at such other place as the Board of Directors may appoint upon notice to all Members, Junior Members and Associates entitled to vote directed to each at his last known post office address, and mailed not later than the first day of January preceding. A report of the financial condition of the Institute and reports of the principal officers and standing committees shall be furnished by the Directors, and presented at each annual meeting.

Sec. 2. Special business meetings of the Institute may be held at such times and places as the Board of Directors may appoint upon twenty days' notice to members given as provided in Section 1.

Sec. 3. At all business meetings of the Institute the presence of twenty-five Members, Junior Members and (or) Associates shall constitute a quorum and the order of business shall be such as shall be fixed in advance by the Board of Directors.

Sec. 4. At all business meetings of the Institute, Members, Junior Members and Associates entitled to vote may vote only in person. Any matter which is presented over the signatures of twenty-five Members, Junior Members or Associates, asking for decision by the membership or which is advocated or opposed by twenty-five Members, Junior Members or Associates present at a business meeting of the Institute on which matter decision is requested, or which the Board of Directors may wish to refer to the membership for decision, shall be sent out to the membership for a vote thereon by letter ballot, accompanied by a statement prepared by the Board of Directors giving the arguments for or against its adoption. Members, Junior Members and Associates residing within the United States of America, Mexico, and Canada, and not in arrears for dues shall be entitled to vote at meetings of the Institute in person or by letter ballot as hereinafter provided.

Sec. 5. All meetings of the Institute other than business meetings shall be held at such times and places as the Board may appoint. Notice of all such meetings shall be given to all Members, Junior Members and Associates by mail or by publication in the monthly magazine of the Institute.

ARTICLE V

Meetings of the Board of Directors

Sec. 1. A regular meeting of the Board of Directors for the election of officers and the transaction of other business shall be held in February of each year, after the annual business meeting of the Institute has adjourned at the place where the latter is held.

Sec. 2. The Board of Directors shall also meet once each month, except in July and August, for the transaction of such business as may come before it.

Any member of the Board of Directors residing more than fifty miles from the City of New York, upon request, shall be paid his actual railway and Pullman fares to and from his residence for attendance at the Annual Meetings held during his term as Director, and similar expenses upon such request shall also be paid to such member for attendance at not more than three other Directors' meetings held in each year of his term.

Special meetings of the Board shall be called by the Secretary at the request of the President, First Vice-President, or any three members of the Board, to meet at any time and place, by notice mailed to the members of the Board at least ten days in advance, or by telegram sent at least five days in advance of the date of the meeting to members in North America.

Sec. 3. At all meetings of the Board of Directors the presence of eight members shall constitute a quorum. In the absence of said quorum at any meeting of the Board, a quorum of the Executive Committee, if present, shall transact such necessary business as is within the powers delegated to that Committee.

Sec. 4. The Chairmen of Local Sections or appointed representatives thereof, and the Chairmen of Technical Committees or Divisions of the Institute, if not members of the Board of Directors, the Counsel of the Institute, and all Past Presidents shall be notified of and have the privilege of attending all meetings of the Board, but without the right to vote. They may participate in discussion, and shall receive the minutes of the Board's meetings.

ARTICLE VI

Directors and Officers

Sec. 1. The Board of Directors shall consist of a President, two Past Presidents, six Vice-Presidents, and fifteen Directors. Eight Directors shall be elected annually by letter ballot: One of said eight shall be designated as "Director and President" and two shall be designated as "Director and Vice-President."

Sec. 2. The officers of the Corporation, who shall also be termed the respective officers of the Institute, shall be a President, six Vice-Presidents, Secretary, and Treasurer. The President shall be that Director who is chosen at the annual election as "President." He shall serve as President for one year, and for the subsequent two years of his directorate be termed "Past President of the Institute." The President shall be ineligible for reelection as such until two years after the expiration of his term as President.

At the first meeting of the Board of Directors after the annual business meeting of the Institute, the Board shall elect a First Vice-President, to hold office for one year, from among the Vice-Presidents of the Board; a Treasurer from among the members of the Board, and a Secretary, who may or may not be a member of the Board, to serve for one year. The Secretary or the Treasurer may be removed from office at any time by a unanimous vote of the Directors present at a special or regular meeting of the Board, due notice having been given in advance that such action is to be considered at said meeting. The Secretary shall receive such salary as may be fixed by the Board of Directors.

The duties of all officers shall be such as usually pertain to their respective offices, together with such other duties as may from time to time be prescribed for them by the By-laws or by the Board of Directors. The Treasurer shall give a bond for the faithful performance of his duties, in a sum to be fixed by the Board of Directors, but at the expense of the Institute. All salaries paid by the Institute shall be fixed in advance by the Board of Directors but no salary shall be so fixed in advance for a longer period than one year.

SEC. 3. In the event of a vacancy occurring in the Board of Directors by death, resignation, promotion by election as President or Vice-President, or for any other reason than retirement at the end of three years' service, the remaining members of the Board shall elect a successor to fill the vacancy and to serve for the unexpired term. A member of the Board whose term has not expired and who is elected President or Vice-President at an annual election shall be considered to have vacated the former office held by him, and the Board shall fill the vacancy as above provided.

SEC. 4. The Board of Directors may, from time to time, in its discretion, appoint any Past Secretary of the Institute "Secretary Emeritus," for such term and with such compensation and duties as the Board may designate.

SEC. 5. The Board may also at its discretion appoint one or more Assistant Secretaries and, or, an Assistant Treasurer, and may define their duties.

SEC. 6. The Board of Directors may appoint standing and special committees charged with such duties as the Board may assign. Such committees shall be directly responsible to and report to the Board; and take only such responsibilities as the Board shall authorize.

SEC. 7. The Board shall have power to appoint and to remove representatives of the Institute on such boards or committees as it may deem wise, and may require from them such reports as it may consider necessary or desirable.

ARTICLE VII

DUTIES OF CERTAIN OFFICERS

SEC. 1. *Secretary:* The Secretary shall keep a record of the proceedings of all the meetings of the Institute. He shall be custodian of the corporate seal, of the minute books, and of all the legal documents belonging to the Institute. He shall have charge on behalf of the Institute of all correspondence. He shall notify all Officers and Directors and all members of committees of their election and appointment; shall issue notices of all meetings of the Board, and of the annual and other business meetings of the Institute; and shall, in calling special meetings of the Directors, specify the objects of such meetings. He shall act as clerk at all the meetings of the Board of Directors and at all meetings of the Institute at which he is present. He shall be custodian of all technical or scientific papers submitted to the Institute for its consideration and shall have general charge and supervision of the editing and proofreading of all material published by the Institute, and of the distribution thereof.

The Secretary may, with the approval of the Board of Directors, employ such persons as are necessary to constitute a clerical force, at such salaries as shall be approved by the Board of Directors or its Executive Committee. He shall be the immediate superior of all such employees.

The Secretary shall do and perform such other duties as may from time to time be assigned to him by resolution of the Board of Directors, and the Directors shall have power, by resolution, to take from the Secretary and to delegate to any other officer any of the duties of the Secretary.

SEC. 2. Assistant Secretaries shall perform such portion of the Secretary's duties as may be assigned by the Board or the Secretary.

SEC. 3. *Treasurer:* The Treasurer shall receive and, under the direction of the Board of Directors, shall disburse all funds of the Institute. He shall keep regular accounts in books belonging to the Institute, which shall be open to any member of the Board of Directors. He shall report in writing at each annual meeting of the Institute, and at every regular meeting of the Board of Directors, the balance of money on hand, and any existing appropriations which may affect the same.

His accounts shall be audited annually by the Finance Committee. The Committee may employ a public accountant to assist it, at the expense of the Institute.

The funds of the Institute shall be paid out by checks signed by the Treasurer and countersigned by the Secretary; in case of absence or disability, the President or First Vice-President may act for the Treasurer, or any Vice-President for the Secretary, but the Treasurer may, at his discretion, place funds of the Institute, not at any time exceeding twenty-five hundred dollars, at the disposal of the Secretary's office for the purpose of paying the smaller current expenses of the Institute and may also establish a pay-roll account. Such funds shall be banked in the name of the Institute and drawn by checks signed by any two of the following persons: Secretary, Assistant Secretary, Assistant Treasurer, or Office Manager. The Treasurer shall examine this account each month and reimburse it by means of checks drawn on the regular accounts of the Institute.

The Treasurer shall be solely responsible to the Institute for all moneys received, whether the same are entrusted to others or not, and may require from any one to whom he entrusts funds a bond, running to the Treasurer personally and taken at the expense of the Institute.

The Treasurer shall do and perform such other duties as may from time to time be assigned to him by resolution of the Board of Directors, and the Directors shall have power by resolution to take from him and to delegate to any other officer any of the duties of the Treasurer.

The Assistant Treasurer shall do and perform such duties as may from time to time be assigned to him by the Board of Directors.

ARTICLE VIII

COMMITTEES

SEC. 1. *Executive:* At the first meeting of the Board of Directors after the annual meeting of the Institute, the Board shall appoint, on recommendation of the President, an Executive Committee of five of its members, three of whom shall constitute a quorum. The Committee shall have all the powers of the Board of Directors when the Board is not in session, including the fixing of salaries of employees, except that it may not: fill vacancies on the Board nor elect or fix salaries of officers; vote extra compensation to officers; vote extraordinary expenses or incur obligations of over five hundred dollars; take action on already invested funds or property rights of the Institute except as authorized in advance by the Board; elect Honorary Members; expel Members; or perform any other act that the Board may interdict.

SEC. 2. *Finance:* The Board shall appoint, at its first session following the annual meeting of the Institute, a Finance Committee consisting of three of its members.

It shall be the duty of the Finance Committee to inquire into and examine the financial condition of the Institute and to consider proper means of increasing its revenues and of limiting its expenses. It shall report at each monthly meeting of the Board, and at other times whenever it shall desire or be directed so to do; and the Treasurer shall at all times furnish it with such statements and information as it may desire. It shall determine, with the approval of the Board, the investment of such surplus moneys as shall belong to the Institute. It shall, at least once in each year, examine the securities belonging to the Institute, and report thereon to the Board. It may, at any time, examine the books and vouchers of the Treasurer and Assistant Treasurer. The Treasurer shall not be a member of the Finance Committee, but shall attend the meetings of the same if requested. The Finance Committee shall present a budget to the Directors at the meeting preceding the annual meeting of the Institute. This budget shall give an itemized estimate of the receipts and expenses of the Institute for the ensuing year. The Directors shall pass on this budget at their first meeting after the annual business meeting of the Institute, modifying it as they consider necessary, and make definite, detailed appropriations for the following year. All bills, accounts, salaries, pay-rolls and claims of every kind against the Institute shall, before being paid, be examined by the Finance Committee and be approved by at least one member of the Committee. The Committee shall not authorize any payments in excess of appropriations. If at any time any of the appropriations seem in danger of being insufficient, the Finance Committee shall report the same to the Directors.

Sec. 3. *Admissions:* The Committee on Admissions shall consist of five members of the Institute, with three alternates, the Chairman and at least one other member being chosen from the Board of Directors, and shall be appointed by the Board, on recommendation of the President, at the first meeting of the Board after the annual business meeting of the Institute. It shall meet at least once a month, but at the Committee's option may waive meeting in any of the summer months. It shall make its own rules of procedure in conformity with these By-laws and subject to the approval of the Board. Its proceedings shall, if required by the Board, be reported to it, but otherwise shall be secret and confidential. No member of the Committee shall propose any candidate and no person shall be considered for election to the Institute within one year after his name shall have been rejected by the Board.

Sec. 4. *Papers and Publications:* The Committee on Papers and Publications shall consist of the Secretary of the Institute, and at least twelve members of the Institute, to assist in passing on all papers offered for publication. They shall be appointed annually by the Board of Directors, on recommendation of the President, and shall have authority, subject to appeal to the Board, to accept or reject papers for publication in the TRANSACTIONS and TECHNICAL PUBLICATIONS of the Institute. The Committee shall formulate a set of rules of procedure which, when approved by the Board, shall have the force of By-laws until revoked or amended.

Sec. 5. *Library:* The Library Committee shall consist of the Secretary of the Institute and four other Members or Associates of the Institute, one being appointed annually by the President, at the first meeting of the Board after the annual business meeting of the Institute, to serve for four years, in conformity with the By-laws of the United Engineering Society, approved by the Board of Directors of the Institute. The Library Committee shall constitute the official representatives of the Institute on the Library Board of the United Engineering Society, and be guided by the By-laws of that Society, as approved by the Board of Directors of this Institute, with full power to act for the Institute within the appropriations allowed by the Board of Directors.

Sec. 6. *Technical Committees:* Technical Committees may be authorized by the Board of Directors, charged with the consideration of such topic or topics as the Board may assign, and having the duty of providing papers and discussions on such topics for the meetings of the Institute. The Board shall designate the officers thereof, who shall hold office for one year or until their successors are appointed. Each Committee may adopt such rules as are necessary for the orderly conduct of its affairs, subject to the approval of the Board of Directors.

ARTICLE IX

NOMINATION AND ELECTION OF DIRECTORS AND OFFICERS

Sec. 1. A ticket of nomination for offices and places annually falling vacant on the Board of Directors shall be prepared by a Committee on Nominations, of which no member shall be, at the time, on the Board. This Committee shall consist of the Chairmen of three Local Sections, one ex-President, and three other members of the Institute, and, on recommendations made by the President of the Institute, shall be appointed by a vote of the Board of Directors at the meeting of the Board held in the month of May, and it shall then proceed to the selection of candidates and the naming of a ticket. In making such selections, the Nominating Committee shall, so far as practicable, distribute the representation on the Board geographically, provided that seven Directors shall be residents of New York City or the territory within a radius of fifty miles of the headquarters of the Institute.

The official ticket thus formulated shall be transmitted to the Secretary in time to be submitted to the Board at its October meeting for its consideration, published in the November monthly magazine, and not later than January first it shall be printed and sent to the membership. Any complete or partial ticket of nominees signed by any twenty-five Members or Associates of the Institute and transmitted to the Secretary by December fifteenth shall also be printed and circulated with the official ticket and over the names and with the recommendations of the nominators, together with an expression of opinion of the Board thereon if deemed expedient.

Sec. 2. The election ballots mailed with the nominations shall contain space for the naming of eight Directors, of which the first shall be designated: "For Director and President," the second and third: "For Director and Vice-President," and the rest: "For Director." They shall be accompanied by small inside adhesive envelopes marked: "Ballot. Do not sign," and outer mailing envelopes addressed to the Secretary and marked "Enclosing Ballots only." Members voting shall not sign their names to the ballot, but seal it in the inner envelope, enclose it in the outer mailing envelope, and write their names on the outside, in the place provided. Cumulative voting shall not be permitted. Letter ballots must reach the Secretary on or before the second Tuesday in February. He shall give them unopened to tellers, to be appointed by the Board of Directors at its January meeting, at least three days before the annual business meeting, marking plainly those from members not entitled to vote, because of geographical location or by being four months or more in arrears for dues. The tellers shall reject unopened any ballots unidentified by the name of the sender on the outer envelope and shall open and count the valid ballots received from members entitled to vote; they shall prepare a report of the voting and hand it to the chairman of the annual business meeting of the Institute. The latter shall declare elected to the respective offices the eligible persons receiving the highest number of votes. The ballots and unopened letters shall be returned to the Secretary at the annual business meeting; the latter shall preserve them one month, and then destroy the ballots which have been counted, and open and destroy the others.

ARTICLE X

Sec. 1. The publications of the Institute shall consist of a Year Book, the Transactions in one or more volumes, a monthly journal called Mining and Metallurgy, a series of individual papers called Technical Publications, and such special volumes and pamphlets as the Board may from time to time authorize. All members in good standing shall receive the Year Book and Mining and Metallurgy, and from the annual dues of each there shall be deducted three dollars, which shall be applied as subscription to said journal for the year covered by said payment. All Members, Junior Members and Associates in good standing shall receive free each year one volume of Transactions, and be allowed to purchase other volumes at special prices fixed by the Board. All publications, including the special volumes and (or) papers accepted for presentation at meetings of the Institute, may be printed and distributed to members under such regulations as to price and distribution as the Board may from time to time adopt not inconsistent with this section.

ARTICLE XI

Divisions—Local Sections—Affiliated Societies

Sec. 1. *Divisions:* Professional groups to be known as Divisions of the Institute and to be organized from its members may be authorized by the Board of Directors. Any member of the Institute may register for membership in any Division in which he is interested and may resign therefrom, provided he be under no financial obligation thereto.

The officers of a Division shall be a Chairman, one or more Vice-Chairmen, a Secretary-Treasurer, and an Executive Committee. The Chairman, Vice-Chairmen, and Secretary shall be ex officio members of the Executive Committee of the Division. All officers of Divisions shall be elected annually by the members of the Divisions, and shall take office at the close of the meeting at which they are elected. They shall hold office for one year or until their successors are elected. A Division shall have the right to make rules for its own government, subject to the approval of the Board of Directors, not inconsistent with the Constitution and By-laws of the Institute. The Board of Directors have the right to amend, annul, or add to these rules. Any Division may raise or collect funds to be expended for its own purposes and may have the entire management and control of such funds in so far as said management or control does not conflict with the provisions of the Constitution and By-laws of the Institute. The Board of Directors shall have the right to recommend the dissolution of any Division subject to approval of such recommendation at any annual meeting of the Institute, in which case the net property of such Division shall be divided pro rata among all members thereof in good standing. Each Division shall furnish the Board of Directors such financial statements of its affairs in such form as the Board of Directors may require.

Sec. 2. *Local Sections:* A Local Section of the Institute may be authorized by the Board at the written request of ten members residing within an appropriate distance of a central point. Only one Section shall be authorized in any one locality or district and the Board shall define the territory of each Section. A Section must include at least twenty-five members. The Board shall have the right at any time to cancel a Section or to readjust its boundaries.

Only Members, Junior Members and Associates of the Institute may be voting members of Local Sections, but Student Associates and others, upon terms laid down by the Local Section and approved by the Board, may become Affiliates or Associates of the Local Section.

All members of the Institute residing within the territory of the Section shall be eligible for membership in such Section. But any such person failing within three months, after due invitation, to become a member of such Local Section, shall thereafter be admitted to its membership and privileges only by such conditions as said Local Section shall determine.

The officers of a Local Section shall be a Chairman, Vice-Chairman, Secretary, Treasurer (or Secretary-Treasurer), and such others as the Section may desire.

It shall be the policy of the Board of Directors of the Institute to contribute from its funds for the necessary running expenses of each Local Section, when and so far as practicable, an amount not exceeding, in each year, twenty-five per cent. of the dues received from the resident members of said Section in said year, but in no case exceeding the sum of two hundred and fifty dollars. Requests for such appropriation shall be signed by the Chairman, Secretary and Treasurer of the Section. If the expenses of a Section exceed the appropriation made by the Board, the difference may be made up by voluntary contributions or local dues from the members of said Section as the latter may determine. The Institute shall not be responsible for the debts of any Section.

Papers presented at Local Sections, and discussions thereon, if reported, shall be the property of the Institute. Such papers shall not be published elsewhere in extenso without permission of the Board. The reading of a paper before a Local Section shall not carry with it the right of publication by the Institute but shall entitle it to consideration by the Papers and Publications Committee.

Local Sections shall be at all times subject to such general regulation as the Board may from time to time elect.

Sec. 3. The Board of Directors may arrange conditions for the affiliation or merger with the Institute of any regularly organized group of engineers, or any engineering society, or any association, or club, whether heretofore or hereafter incorporated, or otherwise, which by aims, constitution, by-laws, or practice, is in accord with the aims of this Institute.

Any society of undergraduates at a technical school, comprising students in any branch of engineering, metallurgy, chemistry, geology, etc., may be recognized by the Board, in its discretion, as an Affiliated Student Society with such rights and privileges as the Board may from time to time by general regulation establish, except that the individual members of such a society are through such affiliation not to become members of the Institute. The affiliation may be dissolved at any time by the Board or the Society.

ARTICLE XII

AMENDMENTS

Sec. 1. The Board of Directors shall have power to make, amend, or repeal these By-laws by a vote of the majority of the Directors present at any regular meeting of the Board at which a quorum is present, PROVIDED, however, that at least eight members shall vote in favor of such making, amendment, or repeal, PROVIDED, FURTHER, that notice of intention to make, amend, or repeal the By-laws in whole or in part, shall have been given at the second preceding meeting, and copies of such amendments shall have been mailed to all the members of the Board at least twenty days before the meeting at which they are voted on and shall have been printed in one issue of MINING AND METALLURGY.

Sec. 2. Any Member or Associate of the Institute in good standing may propose amendments or additions to these By-laws by sending such in writing to the Secretary. The proposal shall be brought before the Board at its next meeting, and, if sponsored by any one member of the Board, shall pass through the course required in the previous section. If the proposal is not so sponsored, it shall be returned to the proposer.

Sec. 3. Amendments or additions to these By-laws may be proposed by sending the same in writing to the Board of Directors so that they are received by them at least sixty days before a business meeting of the Institute, and provided they be endorsed by the signatures of at least twenty Members or Associates in good standing at the time.

It shall be the duty of the Board of Directors to have such proposals printed in the next succeeding issue of the monthly journal, and to provide proper facilities for a vote of the membership thereon by letter ballot before the said business meeting. If favored by a majority of the ballots cast, the amendments shall be reported at the said business meeting as adopted, and shall thereupon become effective.

Cycles in Metal Production*

By D. F. Hewett,† Washington, D. C.

(New York Meeting, February, 1929)

Although most persons will agree that an individual or a nation can profit from the experience of other individuals or nations, there is always room for debate over the degree of similarity of their past and present conditions and of their future problems. Noting the increasing interest that Americans are showing in our mineral problems, I have felt that we could profit from reviews of the experiences in mineral exploitation acquired by the industrial nations of Europe. Whether we can learn much from Europe's experience in exploiting such minerals as petroleum may be debatable, but certainly, with a record of metal production extending back nearly 3000 years, Europe should hold much of value for America.

A trip to Europe in 1926 permitted me to visit 28 mining districts, of which about half are now or have been in times past outstanding sources of several metals. The districts range from England to Greece and from Spain to Poland. To summarize briefly my impressions of these mining districts, as well as such reactions to the thought and life of Europe as so short a visit permits, I shall only venture the opinion that Americans can review with great profit the ideas and actions of Europeans, with regard not only to the production of metals but to many expressions of life. I believe that America has made and is making new contributions to the art of living, as well as to the arts of mining and metallurgy, but in so far as Europe is and has been a laboratory of economic, social and political experiments, the results she has attained deserve the thoughtful attention of America. I have come to believe that many of the problems that harass Europe lie in our path not far ahead. I have therefore hoped that a review of metal production in Europe in the light of its geologic, economic and political background may serve to clear our vision with regard to our own metal production.

One can not examine the literature of national metal production without being impressed that the countries of continental Europe, especially Germany and Austria, have shown greater interest and devoted more time to research into the record of metal production than Great

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* Published by permission of the Director, U. S. Geological Survey

† Geologist, U. S. Geological Survey.

Britain and America, although in Great Britain increased interest since the World War is fast making up for an earlier deficiency. This interest has expressed itself in many ways, notably in exhaustive historical summaries of activities in mining districts, studies of the relation of production to price, and estimates of reserves. Is it significant, however, that the best summary of production of copper in England that I have been able to find was prepared by a German and printed in the Prussian annual review of mineral production?

In making these reviews, the Germans especially have given most attention to the unit which we know as a mining district. Whatever of value may be attached to statistics of national production, the data that concern a district are more basic. As a geologist, I feel partial to such data, because commonly a district is essentially a geologic unit, and geologic features determine the amount of the metals that may be expected to come out of it. National statistics are necessarily aggregates of district statistics.

Obviously, in a brief research I have been able to consider only a few sources of the metals in Europe, perhaps 50 out of a total of many hundreds. In order that the conclusions may have value, however, I have chosen districts which have been important sources during some epoch of the last 300 years or more.

In making this review, I have had several objects. One object is concerned with the behavior of the different geologic types of metal deposits under exploitation. With its long record of production, Europe should be able to indicate the behavior in depth of many varieties of ore deposits. The review should also yield an opinion as to the state of exhaustion of the ore deposits that have been important in the industrial history of Europe. In the third place, a comprehensive appraisal of our own mineral reserves should recognize a relation between past and future production. It seems to me that there are two kinds of reserves. One kind, which may be called "tangible," is the part reasonably in sight. This part is closely related to that which has already been extracted, and the records of the past in geology, technique and cost are trustworthy guides concerning its availability. The other kind may be designated "intangible" and includes all the materials that, from one or another process of geologic reasoning, may be considered to exist. For the intangible reserves, the records of the past afford few clues, and there is a wide field for speculation. I need only add that no sharp distinction separates the two kinds.

Another object of this review is concerned with the attitude of the different national groups toward mineral exploitation. There should be much for Americans to learn from Europe's long experience as a guide to a national point of view that may be expressed as a national mineral policy.

Factors of Metal Production

As I see it, the causes that lead to metal production are multiple—so numerous that I can almost persuade myself that their number exceeds those underlying any other activity that yields tangible goods. Man must eat food and wear clothes, and the agricultural arts must satisfy his needs in those respects. But even in our modern world, with its diverse sources of foods and textile materials, man's unit consumptive capacity for these products does not greatly change from one year to another. On the other hand, his consumption of metals grows with the increasing complexity of civilization and it seems now as if his appetite would never be satisfied.

Necessarily any precise interpretation of the production record of a district must contemplate the possible effect of many influences. I shall enumerate only those groups of factors which seem most important to me.

Geology

First, there are the geologic factors, which are concerned with the minerals present; their number and kind, which determine whether the problem of recovery is simple or complex; the degree of their concentration or dissemination; their border relations; the shape and extent of the recognizable masses. These geologic factors seem to determine how much of each metal may be ultimately recovered.

Technology

Second, there are the technical factors of mining, treatment and refining. A review of these leaves a vivid impression of the labor involved in their improvement but they necessarily yield cumulative benefits. Until the development of the steam pump about 1760, and the steam hoisting engine some years later, progress in mining technique was so slow that rates of metal production were held at figures that look insignificant now.

Economics

The third group of factors that affects rate of production are economic, and among these factors cost and selling price are outstanding. Cost depends upon the price of all that enters into mining and refining, and selling price depends upon competition and all the man-made devices for helping or hindering sale, such as taxes and tariffs. Since 1800 the trend of prices for the common metals, measured not only by monetary units but by the cost in human effort, has been almost steadily downward in all the countries considered. From 1885 to 1900, prices for the metals, expressed as money, reached the lowest figures for the period considered, but expressed in purchasing power, they have been lower in post war years.

Political

The fourth group of factors that affect metal-production curves are political or lie between politics and economics. As we all know quite well from recent experiences, political disturbances, such as wars and revolutions, withdraw men from productive labor and cause declines in production in combatant countries, and by creating demand and higher prices, stimulate production in noncombatant countries. In making this review I have noticed that each war, in addition to taking its toll of human lives, leads to abandonment of mines, some for all time. Wars are relatively brief, however, and even more important are the relations of the state or ruling dynasty to the ownership of minerals, for these exert cumulative influence over long periods.

The task to which I have set myself, therefore, is the interpretation of the curves of production from the outstanding sources of the metals in Europe during the last 300 years, but particular emphasis is placed on the last 125 years, which essentially corresponds to the period of the industrial revolution. Obviously, to interpret with great confidence the curves of production for the 50 or more metalliferous districts which have been reviewed would demand a most exhaustive acquaintance with all the factors of the four groups outlined, in so far as they have affected each district. I make no pretense to such acquaintance, but from the study I have been able to make I have placed particular emphasis first upon the geologic background of the districts and next upon the group of technical factors. Of the economic and political factors I have considered some for a few districts and a smaller number for most of the districts. Such conclusions as I express represent rather confident opinions. My present purpose is to outline a field for study rather than to offer positive conclusions.

Review of Metal Production of Europe

The metals here considered include iron, copper, tin, lead, silver and mercury. It is my purpose to try to trace the trend of production in outstanding districts rather than national trends, which will be mentioned only here and there. Throughout the nineteenth century the tendency of production in most countries has been almost steadily upward, but, as I shall show, this has been accomplished widely by adding new sources as the old sources became depleted—a feature that has characterized the development of the metal industries, as well as the oil industry, in the United States.[1]

In offering production curves of districts in Europe, it is well to offer a word of caution about production graphs in general. No graph is foolproof, and one given to special pleading can so choose the scales of his graphs as to mislead even a careful observer, especially if they cover a long period of time. A given disturbance, price or political, may have the same proportionate effect on the output of two districts, but the district with the larger output shows a more impressive effect on the graph. For purposes of comparison, I have added a few typical graphs of American districts.

Iron

A review of the production of iron in Europe shows most interesting shifts in dominant sources of ore.[2] Much of the Roman conquest was determined by sources of the metals, and wherever Rome penetrated the sources were exploited. Probably during Caesar's campaigns, Elba and northern Italy were the outstanding sources of iron in the world.[3] Small quantities of iron were produced in many places during the Middle Ages, but at the time when the record becomes clear, about 1750, Sweden yielded 65,000 tons of iron annually, or 40 per cent. of the world's production, and Great Britain only 17,500 tons. It seems clear that there were many sources at this time, however, and iron was produced wherever there was iron ore and abundant timber for charcoal. The influence of the mineral fuel, coal, began to be felt about 1618, when iron was first smelted in England with crude coal, but it became dominant about 1735 when the art of coking coal began. From then on, production began to be controlled by the proximity of coking coal to iron ore rather than the proximity of timber for charcoal to ore. About 1800, when the world's output of pig iron was 540,000 tons, Sweden produced 30 per cent., Great Britain about 35 per cent., and Russia 20 per cent. The next 70 years marked the great rise of iron production, and throughout this period Great Britain produced about half the world's supply. Not until 1890 was it surpassed by the United States, and in 1903 it yielded place also to Germany.

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[1] The most comprehensive summary of metal production known to the writer is that of A. Rossing: Geschichte der Metalle (p. 274), Berlin, 1901. In addition to summarizing the history of mining and technique for 57 metals, it presents tables of world production by 5-year periods for most of these and tables of prices. The statistics are uncommonly accurate as far as presented but there are several important omissions from several of the tables. Other similar summaries are by Zippe: Geschichte der Metalle, Vienna, 1857, and Neumann: Die Metalle, 1904.

The tables of production in Mineral Industry, New York, 1893, assemble the records of existing statistical bureaus of European countries, only a few of which were started earlier than 1850.

[2] L. Beck: Geschichte des Eisens, 1903. (5 vol.)

M. Roesler: Iron Ore Resources of Europe. U. S. Geol. Surv. *Bull.* 706 (1921).

[3] H. K. Scott: The Mines of Elba. *Jnl.* Iron and Steel Inst. (1895) 1, 141.

Now, it is worthy of note that during the nineteenth century the production of pig iron steadily rose in each country—in Great Britain, 50 times; Germany, 300 times; France and Belgium, 20 times; Austria-Hungary, 40 times; Russia, 30 times; Sweden, 10 times; and the United States, 700 times.[4] A brief review of the sources of production of iron ore will show what outstanding sources have failed during the extraordinary development of iron production in the last 150 years. Several countries have successively attained dominance in iron production, and the same tendency may be noted among the contributing districts.

Great Britain.—For its area, no country possessed important sources of iron ore and coal so widely distributed as Great Britain 200 years ago.[5] The bedded Carboniferous ores of South Wales were the principal British source from about 1750 to 1850. Production reached the peak in 1871, but recently these ores have all but passed out of the picture. About 1855 production of the bedded Jurassic ores of the Cleveland district began. The peak for this district was passed in 1883, when it supplied 40 per cent. of all British ore; the peak of total British production was passed in 1882. As late as 1900 the Cleveland district produced 57 per cent. of the total, but recent production there has been one-third the peak and 21 per cent. of the total. Production of the similar Jurassic ores of Northamptonshire and Lincolnshire began later but has risen almost steadily to date. In 1918 these ores made up half the total; in 1925, 64 per cent. Two other British sources of iron ore deserve mention—the hematite ores of Cumberland and Lancashire, and the Scottish bedded Carboniferous ores. Though the hematite ores had long been a source of iron, production of these ores rose steadily through the nineteenth century to the peak in 1883; since then it has steadily declined to one-third. These are the principal low-phosphorus ores of Great Britain. The Scottish Carboniferous ores, discovered only in 1801, attained the peak of production in 1880, when they made up 15 per cent. of the British total. The recent production of these ores is scarcely one-seventh the maximum (see Fig. 1).

Of the sources of ore mentioned only the hematites of Cumberland and Lancashire are not bedded; these deposits assume many simple and irregular forms along or near faults. In general, their local distribution much resembles that of the oreshoots of the Leadville district, Colorado. In all the other districts, the unweathered ores are carbonate of iron, and the iron content of the crude ores ranges from 24 to 36 per cent. Residual limonites, as we know them in the belt from Pennsylvania to Georgia and Alabama, are found in many districts but in small quantity only.

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[4] L. Beck: Geschichte des Eisens, **5**, 896 (see table).

[5] See especially J. D. Kendall: Iron Ores of Great Britain and Ireland, 1893, which contains complete production tables for each district; also, R. Hunt: British Mining, 1884; W. Cunningham: The Growth of English Industry and Commerce. (2 vols.) 1910, 1912, 93; L. Beck: Geschichte des Eisens, **3**, 655. Imperial Mineral Resources Bur. *Special Repts., Iron Ore* (1920) 8 vol.

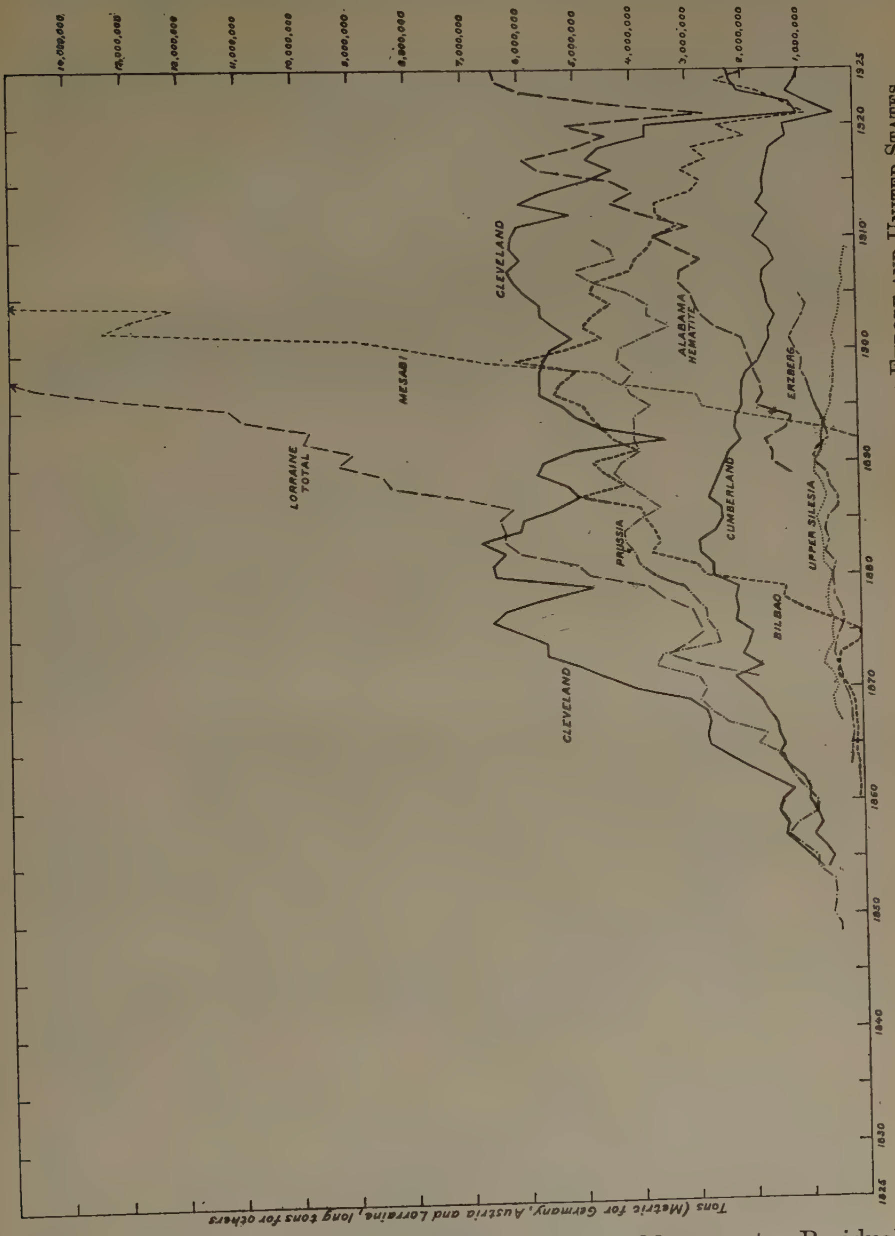

FIG. 1.—ANNUAL IRON-ORE PRODUCTION FROM OUTSTANDING DISTRICTS IN EUROPE AND UNITED STATES.

They were once important in Wales but are now negligible. It is noteworthy that veins of siderite and masses of pyrite and magnetite have yielded very little of the British production.

None of the British ore fields is exhausted. The successive rises and declines reflect competition and relative exhaustion at prevailing prices, and an increase of 50 per cent. in the price of iron ore or pig iron would bring all the districts back as outstanding sources of ore. Finally, concerning Britain's position in iron, it seems clear that at the present ratio of prices, the peak of production of iron ore was passed in 1882. Imports of iron ore began about 1870, and recently have averaged about one-half of the domestic production. The iron content of the imported ore, however, is almost equal to that of the domestic ore.

Germany.—In Germany, as in Great Britain, iron ores are widespread, but during the last 150 years Germany lagged 40 years behind Great Britain in attaining similar annual production.[6] In contrast with Great Britain, only a few of the German districts show distinct declines thus far.

About 1800 the residual limonites of Upper Silesia appear to have yielded more than half the production of German iron. Production of ore rose steadily to 800,000 tons in 1889, and has steadily declined to one-fifth this amount in recent years. On the other hand, production from the iron carbonate veins of Siegerland, which also contain some lead and zinc, rose steadily to 2,500,000 tons in 1907. Other than the bedded ores of Lorraine, the remaining important source of iron ore is the Lahn and Dill valleys, which contain bedded hematite and secondary limonite in limestone. The annual production from this source for the last 40 years has ranged from 800,000 to 1,000,000 tons.

France.—Although iron ores are widespread in France,[7] fewer districts than in Germany are large producers. Outside of the bedded oolites of Lorraine the ores in only two districts deserve mention. As recently as 1913, the Silurian bedded carbonates of Normandy yielded 800,000 tons of ore, or 4 per cent. of the total from France, but in 1927 the output exceeded 2,000,000 tons. The beds and veins of carbonate of iron in the eastern Pyrenees have yielded 300,000 to 400,000 tons annually for 20 years, but the present epoch of exploitation seems to mark the peak.

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[6] L. Beck: *Op. cit.*

G. Einecke and W. Kohler: Die Eisenerzvorrate des Deutschen Reiches. *Archiv. für Lagerstatten Forschung*, Preus. Geol. Land (1910).

Ley: Zur Geschichten und ältesten Entwicklung der Siegerlander Stahl und Eisen Industrie (1906).

Handbuch des Oberschlesischen Industrie-bezirks, Festschrift Oberschl. Berg. und Hutten Verein (1913) **3**, 690.

[7] Imperial Mineral Resources Bur. *Special Repts.*, *Iron Ore*, (1922) **6**, 35.

M. Roesler: *Op. cit.*, 55.

The outstanding iron-ore field of Europe is that containing the Jurassic oolite of Lorraine and Luxemburg; that portion of the field which lay in Belgium was exhausted years ago. As Roesler has shown, the rate of increase in production from the district since 1870 is almost as great as that of Lake Superior, and it seems that the peak of production is still years ahead. Although in 1871 this district yielded only 6 per cent. of Europe's total production, in 1913 it yielded one-half. In the years preceding the World War it yielded 90 per cent. of the iron ore of France, and 70 per cent. of that of Germany without Luxemburg. As the known reserves contain more than 5,000,000,000 tons, this rate of production could be maintained for about 100 years. On a competitive basis, without political interference, this district could probably supply 75 per cent. of the iron consumed in Europe for 50 years.

Belgium.—The iron-ore production of Belgium served to establish a smelting industry which has long survived the local sources of ore. Some deposits were worked many centuries ago, and there is a complete record of production back to 1836.[8] The ore was obtained principally from the weathered outcrops of sulfide veins, and the oxidized ores are now exhausted. The output of domestic ore passed the peak in 1865, when 1,000,000 tons was produced, but recently it has been scarcely one-seventh this amount. The smelting industry survives on ore imported to the extent of 7,000,000 tons annually.

Austria.—Records of the exploitation of the iron-ore deposits of Erzberg in Styria extend back to 931, and the annual output shows a surprisingly uniform increase for the last 200 years to the maximum in 1913.[9] Fig. 1 represents the output of the province of Steiermark, but the mines of Erzberg supply almost all the ore produced in the province. In 1880, the output of Steiermark was 70 per cent. of that of Austria, and for 50 years it has exceeded 50 per cent. This record of steady growth has greater interest in view of the fact that the reserves will last more than 100 years longer at the present rate of production.

Spain.—One district in Spain has occupied an outstanding position in that country as well as in Europe.[10] The limonite deposits of Bilbao represent the weathered product of siderite which has replaced Cretaceous limestone. Production began about 1850, but after the Carlist war, in 1874, it rose rapidly to its peak of 6,000,000 tons in 1899. For 75 years this district supplied more than half the Spanish production, and in the height of its productivity more than three-fourths. Since then there has been steady decline to one-third the peak, and qualified observers believe that the recent grades of ore will be exhausted in 15 years. The fate of these deposits attracts interest because they have been the principal source of high-grade ore imported into Great Britain to supplement the domestic low-grade ore.

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[8] Stat. des Mines, Min. Car. et Usines met. Belgium.

A. Delmer: La question du minerai du fer en Belgique. *Annales des Mines de Belgique* (1913) **18**, 325, 448; (1912) **17**, 853, 940.

[9] A. Aigner: Mineralschätze der Steiermark (1905) 228.

H. Bauerman: The Erzberg of Eisenerz. *Jnl.* Iron and Steel Inst. (1907) **75**, 27.

[10] E. Dann: Die Eisenerzlager und die Eisen industrie von Bilbao. *Stahl u. Eisen* (1913) **33**, 1181, 1232; *Boletin Minero* (Bilbao) (1926) **5**, No. 49, 9.

Sweden.—There is an uncommonly good record of Swedish iron-ore production back to 1300, and the mines appear to have been the outstanding European source during the Middle Ages.[11] As late as 1740, Swedish mines yielded 40 per cent. of the world's iron. The Swedish record is interesting because each of four outstanding sources of ore and the total for the country show an almost steady increase in output for the last 100 years. The present principal mines, at Kirunavarra, were first worked only about 1900, but their production has increased almost steadily since then. The Swedish deposits appear to be either segregations from intrusive rocks or of contact-metamorphic origin and are the most productive examples of the types.

Russia.—The trend of iron-ore production in Russia closely resembles that of Sweden, although the Russian deposits are more widespread, show greater variety, and probably contain larger reserves.[12] The deposits that have been most productive recently include hematite masses near Krivoi Rog, in South Russia, and in the Ural region. Too little is known concerning the annual yield over a long period to determine the state of exhaustion of any of the deposits, but under favorable political conditions production would probably continue to rise for some years in most if not all the districts.

Copper

We have fairly good records of the sources of our copper supplies since 1800,[13] but for the period prior to 1800 good records of production are available for only a few districts, notably Cornwall and Mansfeld.

The records of copper and silver production from Mansfeld[14] extend back to 1688, and we know that the mines were worked as early as 1199. This district has the amazing record of almost constant rise in annual production for at least 250 years. The output for 1913 is the highest attained, but the years to come will probably witness still higher production. For the last 40 years the district has yielded more than 80 per cent. of the German total. (Fig. 2.)

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[11] H. Lundbolm: Iron Ore Resources of the World (1909) 590.
Also Sveriges Officiella Statistik, *Special Studies* (1915) 51.

[12] Imperial Mineral Resources Bur. *Special Repts.*, *Iron Ore* (1922) 6, 157.

[13] C. E. Julihn: Summarized Data of Copper Production: U. S. Bur. Mines *Econ. Paper* 1 (1928).
Brown and Turnbull: A Century of Copper (1898) 11.

[14] H. Hoffman: Gedenkschrift, 1925. Mansfeld, A. G., fur Bergbau und Hutten betrieb, 1925.
Schrader: Der Mansfelder Kupferschiefer Bergbau. *Ztsch. f. d. Berg. Hutten u. Salin. Pr. St.* (1869) (17) 251.
Die Geschichte des Mans. Kupfer. Festschrift, Eisleben, 1900.

Records of production of the Falun district, in Sweden, continue back to 1633, and the mines were worked 500 if not 1000 years earlier.[15] About 1650 this district appears to have been at its peak, and was the principal European source of copper, followed in order by Cornwall and Mansfeld. A hundred years later Falun had declined to 40 per cent. of its peak, and other Swedish and Norwegian districts had arisen. At this time Cornwall and Mansfeld had nearly equal outputs and furnished most of Europe's supply.

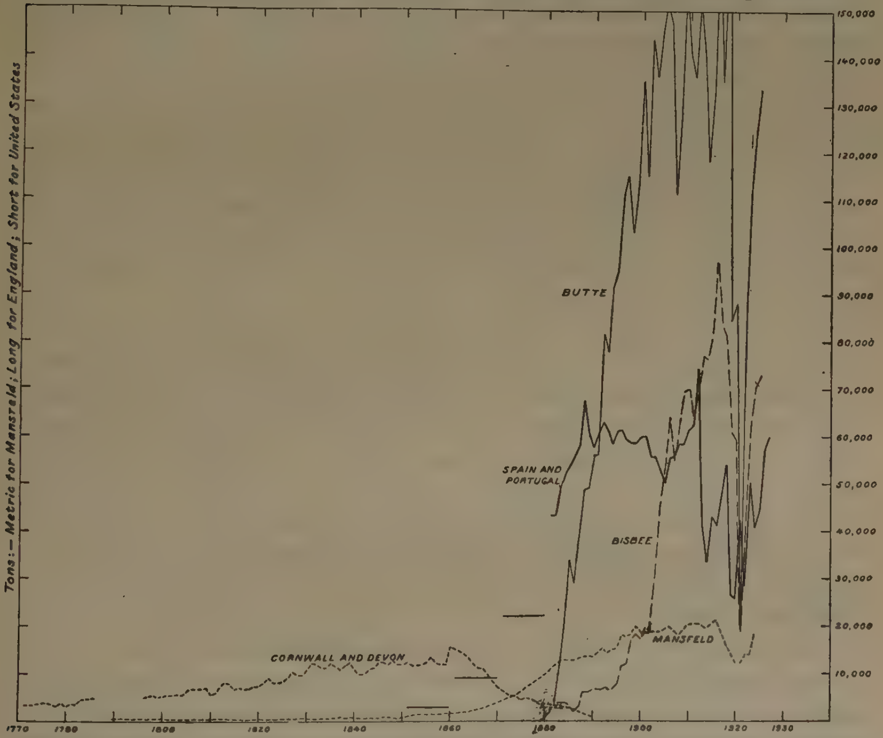

FIG. 2.—ANNUAL COPPER PRODUCTION FROM OUTSTANDING DISTRICTS IN EUROPE AND UNITED STATES.

Cornwall was the scene of tin mining 800 years before Christ, and there is a good record of the activities there since 1156.[16] When placer mining for tin gave place to lode mining the copper associated with the tin was thrown away, but in 1580 the visible supply of copper ore justified the establishment of smelters at Swansea, which have continued to our day. Copper production from Cornwall and Devon rose very steadily to the maximum in 1861 but has declined steadily since then and is now nearly extinct. From 1820 to 1840 the Cornish output was 45 per cent. of that of the entire world. (Fig. 2.)

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[15] Sveriges Officiella Statistik, *Special Studies* (1912) 7.

[16] R. H. Lamborn: Metallurgy of Silver, Lead and Copper, 228. London, 1878.

L. Odendall: Die Entwicklung der Englisher Kupfer produktion, *Ztsch. f. d. Berg. Hutten u. Salin, Pr. St.* (1910) 65.

H. Dewey: Copper Ores of Cornwall and Devon. *Special Reports on the Mineral Resources of Great Britain* (1923) **27,** 74.

In connection with any review of the European copper situation, the part played by the Swansea smelters is interesting.[17] Although there were copper mines in Wales when these smelters were established in 1582, Cornwall contributed most of their ore for 250 years. There appear to have been no imports of copper ore into Britain until 1826, when it was first brought from Chile. Imports rose rapidly, so that by 1843 the copper in imported ore exceeded that in Cornish ore. During the period 1875 to 1895, Swansea was the world's greatest copper-smelting center, owing apparently to the advantage of fuel supply and superior technique, but since 1895, when domestic production ceased, the industry has survived wholly on imported ore.

The Rio Tinto and Tharsis districts, in Spain, were a source of copper and probably of lead and iron 800 years before Christ, as well as during the Roman occupation.[18] The area appears to have been nearly idle throughout the Middle Ages, and its modern history begins about 1825. From 1850 production increased rapidly but compared with that from other European sources has shown wider fluctuations. From 1881 to 1890, the output of copper from these districts was twice that of the rest of Europe and one-fourth that of the entire world. The peak of production was passed in 1912, but under favorable conditions may be exceeded in the future.

The competitive position of Cornwall, Mansfeld and Rio Tinto is obscured by the complexity of their ores. In the Cornwall veins copper tends to disappear in depth and to be succeeded by tin, so that the present sources of tin were once worked for copper. The bedded ores of Mansfeld have a complex composition and in addition to copper yield considerable silver. The great pyritic masses of Rio Tinto yield iron and sulfur as well as copper, and the output depends upon the market conditions in respect to each of these products.

Tin

The district of Cornwall and Devon, in England, has been outstanding as a source of tin for at least 1000 years. There are several districts on the continent, notably in Germany and Spain, but their output has never been more than a small fraction of that from England.[19]

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[17] G. Curtis: The Smelting Industry of Copper in the Swansea District from the Time of Elizabeth to the Present Day. London, 1881.

[18] L. DeLaunay: L'industrie du Cuivre dans le région de Huelva. *Annal. des Mines* (1889) *Mem.* 16, 427, 515.

The debate among historians concerning the earliest work in Cornwall may be ignored, but there is authentic almost complete record of annual production back to 1156, or for 770 years. Until about 1500 all the tin oxide produced was washed from stream and beach alluvium. The next 200 years witnessed the transition to lode mining, and since 1700 very little alluvial tin has been recovered. Blasting with gunpowder was introduced from Germany about 1670, and with the application of steam to pumping and hoisting from 1750 onward shafts became steadily deeper.

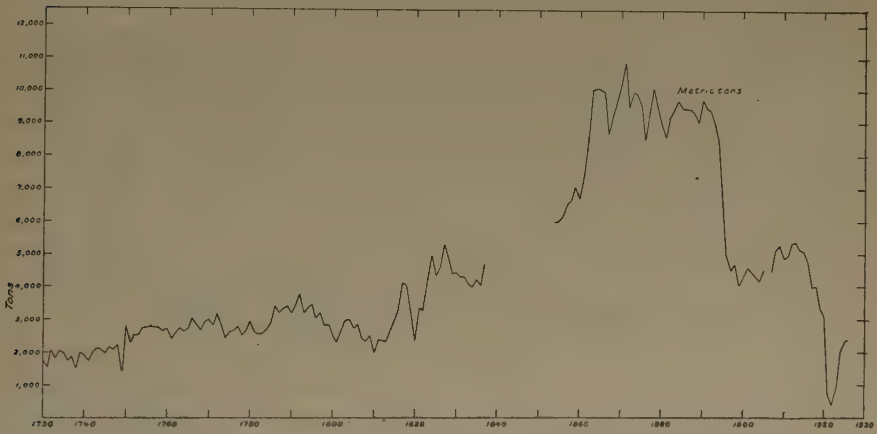

FIG. 3.—ANNUAL TIN PRODUCTION FROM CORNWALL AND DEVON.

The Dolcoath shaft, the deepest metal mine in Great Britain, is 3240 ft. deep, but since the World War nearly the lower 1000 ft. has been under water.

The record of production shows a surprisingly steady rise to about 1820, then an abrupt rise to the peak in 1871, then 20 years of uniform production, and since 1890 a nearly uniform decline. Long before the peak was reached, ore was imported for smelting, with the result that in 1913 tin recovered from imported ore was four times that recovered from domestic ore. (Fig. 3.) In addition England imports considerable tin metal.

From 1800 until the World War the maximum price has been about two and one-half times the minimum. The high prices have coincided with wars and similar disturbances.

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[19] G. L. Lewis: The Stanneries. *Harvard Econ. Studies* (1908) **3**, 252.

N. M. Penzer: Tin Resources of the British Empire, 18. 1921.

E. Z. Reyer: Eine geologische-montanische-historische Monographie, 248. Berlin, 1881. Also translation by R. Symons: History of Tin. Truro, 1881.

Lead

The trend of production of lead from time to time has been closely related to the silver and zinc content of the ores. Thus, the production of lead in the Mediterranean regions before Christ seems to have depended wholly upon the recovery of silver, most of which was obtained by cupelling lead bullion.[20] That the conquests of Rome were closely related to the search for lead and silver is indicated by the fact that most of the districts west of the Rhine and south of the Danube, as well as in England, that have become important sources in recent times were known and exploited by the Romans largely to recover silver. Most of the districts in central Europe that have become notable producers were discovered later. Only England, Spain and Germany will be considered here.

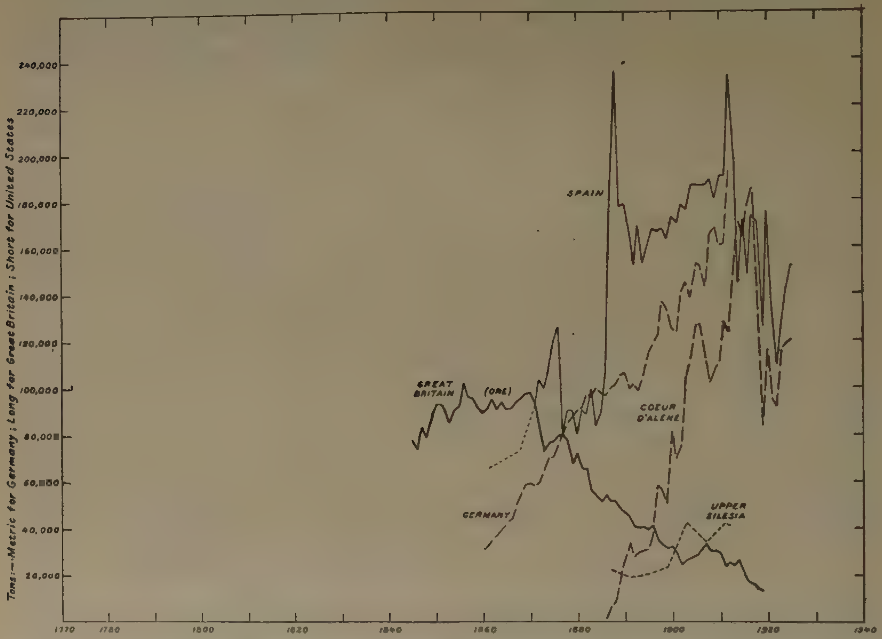

Fig. 4.—Annual lead production from outstanding districts in Europe and United States.

Many lead deposits in England were worked by the Romans, and some believe that the Roman wall that still stands near the Scottish border, as well as most of the Roman roads, were constructed to protect and serve the lead mines.[21] Mining lagged through the Middle Ages but took on new life in the time of Elizabeth. Several writers attach much significance to the importation of German miners about 1580. For the entire period from 1700 to as late as 1850, England was the source of at least half the world's lead. (Fig. 4.) Not less than eight districts contributed, but that of Northumberland and Durham yielded nearly half the total. The production of ore culminated in 1856 and recently has been scarcely one-eighth the maximum.

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[20] L. DeLaunay: Le mines de Laurion dans l'antiquité. *Annal. des Mines* (1899) *Mém.* 16, 1.

L. DeLaunay: Histoire de l'industrie minière en Sardaigne. *Annal. des Mines* (1892) **1,** 511.

T. A. Rickard: Notes on Ancient and Primitive Mining and Metallurgical Methods. *Eng. & Min. Jnl.* (1926) **122,** 649.

W. H. Pulsifer: Notes for a History of Lead. New York, 1888.

The Spanish lead deposits, especially those near Cartagena, had been worked by the Romans, but production declined and was negligible for many centuries. In 1825 prohibitions against the operation of the mines were revoked and the industry grew rapidly, so that by 1870 the Spanish production exceeded that of England. The peak was reached for the silver-bearing ores of Cartagena in 1898, and the output has steadily declined since.[22] The veins of Linares were exploited more slowly, and their production attained the peak in 1907. The total Spanish production seems to have culminated in 1912. When English lead mines began to fail, their owners turned to Spain, with the result that many of the largest deposits are still owned by them. The French and Germans also own important mines.

Germany possesses many productive lead districts, notably the Eifel in Rhenish Prussia, and the Harz.[23] Upper Silesia, an outstanding source of lead, as well as zinc, is now largely ceded to Poland. Production in the Eifel seems to have passed its peak in 1884, but Rammelsberg, in the Harz, increased almost steadily until 1914. The trend of production in Upper Silesia has been almost uniformly upward for 200 years.[24]

Silver

Much is known concerning the sources of silver as far back as 600 years before Christ. The lead mines of Laurium, in Greece, were the principal source of silver for 300 years. From the slag dumps it has been estimated that this district, from 600 to 300 B.C., yielded 2,100,000 tons argentiferous lead containing about 130 oz. silver to the ton, or a total of

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[21] Special Reports on the Mineral Resources of Great Britain. *Mem. Geol. Surv.* (1921) **21, 22**; (1923) **25, 26**.

[22] R. Pilz: Die Erzlagerstatten von Cartagena in Spanien. *Ztsch. f. prak. Geol.* (1908) **16**, 177.

[23] J. A. Phillips and H. Louis: A Treatise on Ore Deposits, 362. 1896.

F. Klockmann: Die Erzlagerstatten des Oberharzes. *Berg und Hutten des Oberharzes*, 43. 1895.

[24] *Handbuch des Oberschlesischen Industrie-bezirks*, Festschrift Oberschl. Berg. u. Hutten. Verein (1913) **2**, 72, 451.

273,000,000 oz., an average of 900,000 oz. a year for those 300 years.[25] Other lead deposits in Spain and Italy were exploited by the Romans for silver. For the next thousand years after the fall of Rome, or until 1400, little is recorded. The fifteenth and sixteenth centuries saw a great growth in production of silver in several parts of Europe, notably in Hungary (Schemnitz and Kremnitz), in the Erzgebirge of south Germany and Bohemia (Freiberg, Annaberg, Joachimsthal, Schneeberg, and Pribram), and in Spain (Guadalcanal). Curiously, most of these districts show considerable similarity geologically, as their deposits are simple veins containing rich silver minerals. Also, the silver of most of them was largely exhausted before 1850.[26] Doubtless the production from many mines in Europe between 1600 and 1800 was affected by the flood of

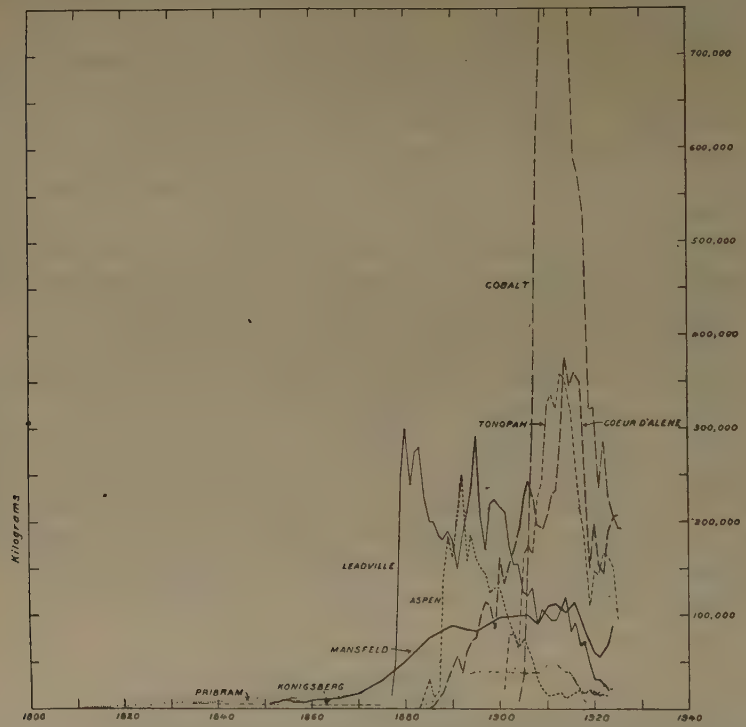

FIG. 5.—ANNUAL SILVER PRODUCTION FROM OUTSTANDING DISTRICTS IN EUROPE AND UNITED STATES.

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[25] Cordella: Le Laurium, Desc. de prod. des mines et usines du Laurium. 1896. See also T. A. Rickard: *Op. cit.*

[26] W. Jacob: Production and Consumption of the Precious Metals. Philadelphia, 1832.

R. H. Lamborn: *Op. cit.*, 236.

silver from the New World—Mexico, Peru and Bolivia—but it has been difficult to trace the influence of this new source on separate districts.

For most of the period since 1800, 90 per cent. or more of the silver produced in Austria has come from the State-owned mines at Pribram.[27] (Fig. 5.) It is noteworthy that the production of this district, exploited since the thirteenth century, rose almost steadily to its peak in 1913, although since 1895 operations have shown a deficit in every year except one. The mines have attained a depth of 4335 ft., by far the greatest depth among the metal mines of Europe.

The veins of Konigsberg, Norway, discovered in 1623 and owned by the State since 1636, have yielded most of Norway's silver since that date. There was a peak of production about 1750, then a decline to zero about 1810, then a nearly steady rise to recent figures of 6000 kg. a year.[28]

The mines of Sala, long the outstanding source of silver in Sweden and during the Middle Ages one of the principal sources in Europe, were worked as early as the sixth century, and there are good records of production since 1520, when the peak, about 5000 kg. silver, appears to have been attained. After a decline to 1610, production again rose almost steadily to 1895 and has since declined nearly to zero.[29]

The peak of production of silver in Germany seems to have been passed when 489,000 kg. was produced in 1892, but viewed broadly the production rose steadily to that amount and has not declined greatly since then. In the Middle Ages most of the silver produced in Germany appears to have been derived from the rich silver veins of the Erzgebirge and the Harz.[30] With their decline, production from the lead and complex copper ores of the Harz, especially Mansfeld, has steadily risen, so that since 1870 Mansfeld has yielded about one-fourth of all Germany's silver. Although production began in 1199, the maximum was not attained till 1916.[31] Similarly, the silver obtained from the lead-zinc ores of Upper Silesia, which were first worked in the twelfth century, has risen steadily to the present time, although it amounts to only 3 per cent. of the German total. On the other hand, production of silver from the classic districts of Freiberg in Saxony, Clausthal in the Harz, and the Eifel in Rhenish Prussia seems to have passed the peak in the later part of the nineteenth century.

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[27] J. Hrabak: Mines and Metallurgy of Bohemia, 331. (In Czech.) Prague, 1922. J. Diviš: The State Silver and Lead Mines of Pribram. (In Czech.) Prague, 1926.

[28] S. Rolland: Memoire sur la géologie de Konigsberg. *Annal. des Mines* (1877) **11**, 391.

[29] Sveriges Officiella Statistik, *Special Studies* (1913) 16.

[30] L. DeLaunay: Traité de metallogénie: gites metallifères, **3**, 276. Paris and Liége, 1913. C. Béranger.

[31] H. Hoffman: *Op. cit.*

The lead deposits of several parts of Spain have been worked since pre-Roman days as sources of silver and lead. The veins of Guadalcanal were the source of large quantities of silver from 1598 to 1635, but these appear to have been long exhausted. Veins near Hiendelaencia, discovered in 1843, yielded about 250,000 kg. silver before they were abandoned in 1866. Outside of these two districts, most of the silver of Spain has come from lead ores, largely derived during the last century first from Cartagena deposits in limestone and then from the Linares veins. The peak of production of Cartagena was reached in 1898, when 149,652 tons of lead ore was produced; that of Linares in 1889, when 118,000 tons of lead was produced.[32]

Europe has only a few representatives of the high-grade silver deposits related to Tertiary intrusive rocks which have yielded such enormous quantities in the Sierran belt of North and South America. The silver produced during Phoenician, Greek and Roman times was obtained largely from lead ores, by cupelling base bullion. The great production of silver in Austria, Germany, Spain, Sweden and Norway through and after the Middle Ages came from the purer and richer ores relatively free from lead and copper, for which a new metallurgy was created. Production from these sources largely passed before 1800, and since then most of the silver has come from the more complex lead and copper ores. Our own production of silver tends to repeat this succession. The great output of the West from 1860 to 1880 came from the relatively pure, rich ore of the Comstock lode and similar ores from Nevada, Utah and Colorado, and the contribution since then has come more and more from the complex copper and lead ores.

Mercury

Four districts have produced most of the mercury of the world for many years. More than the deposits of any other metals, these possess outstanding similarities in mineralogy and form, if not in age and local geologic setting. A comparison of the production from the three principal sources in Europe—Almaden in Spain, Idria and Monte Amiata in Italy—with that from California shows an impressive contrast.

Probably production began first in Almaden, about 400 B. C., but some mines of Monte Amiata also were worked before Christ. The record of production from Almaden is known since 1500 and shows an almost constant rise to the peak of 1888, except for temporary declines during the epoch of the Napoleonic wars.[33] The deposits have been owned by the State for nearly 600 years and have been exploited by it for nearly 400 years. Recent production has averaged about 60 per cent. of the peak of 1888. The grade of the ore, about 8 per cent. of mercury, has long exceeded that of any other important source, and even though the ores are being mined 1300 ft. below the surface, the known reserves are large.

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[32] L. DeLaunay: *Op. cit.*, 3, 73, 130.

[33] H. Sampelayo and others: Mines d'Almaden, Internat. Geol. Congress (1926) 107.

The deposits of Idria were discovered in 1490 and were the property of the Austrian State from 1580 to 1918.[34] There are exceptionally complete records of operation since 1780 (Fig. 6). One peak of production occurred from 1785 to 1800, but this has been exceeded recently. In some informed quarters it is believed that the known reserves will not last many years. When good records became available, 140 years ago, the ore mined yielded 16 per cent. of mercury, but this declined steadily to 2 per cent. 60 years ago, and recently it has been only 0.65 per cent., or 13 lb. to the ton.

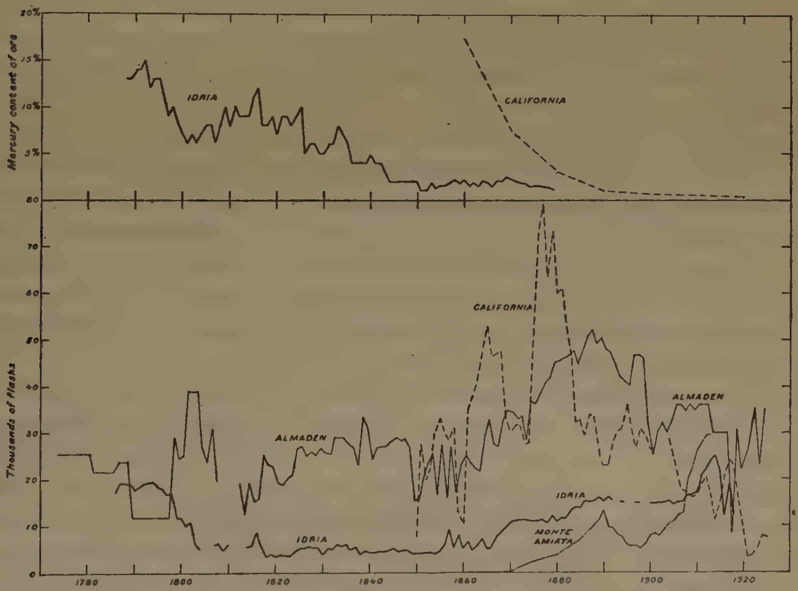

FIG. 6.—ANNUAL MERCURY PRODUCTION AND CONTENT OF ORE FROM OUTSTANDING DISTRICTS IN EUROPE AND UNITED STATES.

The deposits of Monte Amiata that have been worked recently were discovered in 1846, and although there have been great fluctuations in output, that of recent years represents the highest attained.[35]

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[34] Das K. K. Quicksilberwerke Idria, Vienna, 1880. Volume published on Tercentenary of State Ownership, 1580–1880.

[35] C. De Castro: Le Miniere di Mercurio del Monte Amiata, 203, 1914.

Our own California deposits were discovered in 1824 but yielded little until 1850.[36] The peak of production was attained 27 years later, in 1877, and since then there has been steady decline. The ore treated early contained 17 per cent. metal, but the tenor declined quickly to less than 5 per cent. and in recent years has averaged 0.4 per cent., or 8 lb. to the ton. The contrast between the production curves of the controlled districts, Almaden and Idria, and our own uncontrolled California is striking.

Review

I shall attempt to interpret the production curves which I have examined, in the light of some of the geologic, technical, economic and political factors referred to earlier.

No one can review the record of European metal mining without being impressed (1) by the shifts in supply from one outstanding source to another during the past 125 years, (2) by the declines in production from most of the outstanding sources, and (3) by the declines in grades of ore treated. Also, by contrast, our American sources show a greater number and greater range of variations in production. Whether other tendencies are impressive or only interesting will depend upon the range and depth of the observer's experience.

Without discussing the causes here, it is impressive to note the shift in the last 300 years in the outstanding sources of iron from Sweden to England, then to Germany, and since the war to France through the transfer of Lorraine. Similarly, districts in Norway, England, and Spain were successively outstanding sources of copper, and in England, Germany, and Spain, of lead. Obviously, declines in production are numerous, but they are particularly noteworthy for most districts yielding copper, lead and iron in England; iron and lead in Spain; silver in Germany.

Geologic Factors

Viewed in the light of their geologic setting, the sources of Europe's metals have been for a long time and are now diverse, as in fact they are the world over. Europe almost wholly lacks representatives of the Tertiary deposits of copper, lead, silver and gold which have made the Rocky Mountains and Cordillera of the two Americas such notable sources of supply. Most of her deposits of these metals belong to earlier epochs, especially the Hercynian. In respect to form, veins or simple lodes have been notable sources of copper, tin and lead in Cornwall and Devon; of lead widely throughout England; of mercury, lead and silver in Spain; of silver and lead in Germany, Austria-Hungary and Norway. Most of these have failed in depth, owing to the fact that the metals were part of a zonal scheme. For those which have failed, it would seem that the supply is exhausted for all time. I think particularly of Cornwall and of Joachimsthal and others in the Erzgebirge.

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[36] F. L. Ransome: Mercury. U. S. Geol. Surv. *Mineral Resources* (1918) Pt. 1, 159.

Europe lacks disseminated deposits of copper due to supergene enrichments such as those in Nevada, Arizona, Utah and New Mexico, which yield a large part of our own supply. The iron ores of Upper Silesia resemble the residual ores of Pennsylvania, Virginia, Georgia and Alabama, but they have not proved an important source over a long period. The supergene limonite of Bilbao is derived from concentrated bodies of siderite and is the nearest approach in Europe to our deposits on the Mesabi Range.

There has been a steady shift from richer sources of iron to the lower grade bedded oxide or carbonate ores, such as are found in Lorraine and several parts of Great Britain, thus showing the degree to which the modern iron industry will sacrifice grade for assured quantity. The Swedish magnetites continue to be the richest ores mined in large quantity, as they yield about 63 per cent. iron. On the other hand, most of the bedded iron ores of Europe contain less than 36 per cent. iron and a considerable output of bedded carbonates from Oxfordshire, England, contains only 24 per cent. iron.

Among the sources of copper, the Cornish ores declined in average content from 12 per cent. in 1770, to 6.5 per cent. in 1860, and doubtless fell still lower in the later years. The pyritic ores of Rio Tinto, which averaged 3.2 per cent. copper in 1884, have declined to 1.6 per cent. On the other hand, in a hundred years the average grade of Mansfeld ores declined only from 2.5 to 2 per cent. copper, and that only in the last five years, but the grade of ores there appears not to be related to depth.

Data for mercury ore have already been given in Fig. 6.

A brief contrast of these figures with the statistics for American metalliferous products may be interesting. The average grade of the entire Lake Superior output of iron ore, which makes up 87 per cent. of our total, is just below 52 per cent. iron, and this average has been sustained for 20 years only by an increase in beneficiation. Our lowest grade of ore, the hematite from Alabama and Georgia, contains 35 per cent. iron. Most of our copper comes from ores that average about 1.2 per cent., and some contain no more than 0.9 per cent. One may well ask how long this tendency to treat lower grade ore will continue. Without attempting to give a definite answer for any area or entering into an exhaustive review here, it seems that most geologists would agree that some metals are widespread in the earth's crust, whereas others are distinctly localized. Such metals as iron, manganese, aluminum, and even gold, are so widespread that supplies will always be obtainable if we are willing to pay the price in fuel, power and labor. Such metals as mercury, lead and chromium, however, seem to be distinctly more localized—that is, they have a different order of dissemination in the crust. For these, we shall much sooner reach the time when successive increments in price will fail to yield appreciable increments in quantity.

Deep Mining

In the field of mining technique, depth of mining is the most interesting feature. The deepest metal mine in Europe is the Anna shaft at Pribram, Czechoslovakia, 4335 ft., and there are four other shafts in the district more than 3640 ft. deep. The Dolcoath shaft, in Cornwall, is 3240 ft. deep. Lodes at Konigsberg and Clausthal have been worked to depths of about 2800 ft. At Andreasburg, in the Harz, a depth of 2602 ft. was attained 30 years ago, but the mine has since been abandoned. The veins of Freiberg and Siegerland have been followed to about 2100 ft., but deep work has recently ceased. Most of the mines in the classic metal districts, such as Joachimsthal, the Harz, Linares, and Almaden, range from 1200 to 1600 ft. in depth, and in most places little or no deeper work seems to be in sight. One can not review the depth of metal mining in Europe without realizing that there is widespread abandonment of deep work or slow progress in depth.

Our own recent work, on the other hand, is carrying exploration to much greater depths. The Central Eureka, Kennedy and Argonaut shafts on the Mother Lode are 4850, 4460 and 4300 ft. deep, respectively; at Butte there are four shafts from 3122 to 3770 ft. deep; in the Coeur d'Alene district, two shafts attain vertical depths of 3400 and 4000 ft. below the surface; in Michigan one inclined shaft attains a vertical depth of 5309 ft., and there are several more nearly as deep. There are numerous districts with explorations from 2000 to 3000 ft. deep.

Obviously production from veins and lodes can be sustained only by continued exploration in depth. A comparison of the rates of sinking deep shafts during the past 25 years in Europe, especially at Pribram and Linares, and in America, at Butte and Coeur d'Alene, indicates that at similar depths we are deepening our shafts two to four times as fast as Europe. While admiring the ingenuity and superb skill shown in attaining these greater depths, one who emphasizes the geologic elements of the problem can hardly avoid the suspicion that the time is not far distant when our deep shafts will have attained the maximum depth of profitable work, even if the distribution of the minerals does not conform to a simple zonal scheme.

Smelting Industry

I shall not attempt to review the trend of technique in the field of smelting, but wish only to point out that one important result of a local metal-mining industry has been the creation of a smelting or refining industry which has survived in a flourishing condition the exhaustion of the local mines. This is noteworthy with copper, lead, tin and iron in Great Britain; lead, zinc and iron in Belgium; and to a lesser degree with lead, zinc and iron in Germany. These countries are well endowed with fuels, especially coking coal. It might even be said that from an economist's point of view such a smelting industry is a device for selling a country's coal or power at a higher price and, to a lesser degree, for selling its technical skill. There can be little doubt that these European smelting industries will survive as long as their coal can be produced competitively. For our own part, most of our smelters lie inland and when their sources of ore fail they will face serious problems. Moreover, at some remote time—how remote I shall not discuss—our seaboard smelters and refiners will depend largely on imported ores.

State Ownership

Of the economic and political factors that have affected metal production, I wish to emphasize two relating to ownership—first, the relation of the State or royal house to ownership, and second, the element of absentee ownership. In Europe from the earliest times metal mines seem to have been the property of the State or ruling family. Thus, we find that the silver-lead mines of Laurium were owned by the Greek Republic and were leased for operation. The same may be said of metal mines under the Roman Republic and Empire. When and in what ways other types of ownership developed I have not traced closely and will not discuss here. When we pick up the record in the sixteenth and seventeenth centuries, we find that the States of Europe owned most of the metal mines—those of mercury at Almaden and Idria, of silver at Konigsberg, Sala, Guadalcanal, and many more. State ownership of many of these mines has continued to the present day, especially in Austria. Of the countries that abandoned State ownership earliest and most thoroughly, England stands foremost. With reference to this feature the map of Europe can be divided into two parts—on the one hand, the regions east of the Rhine and north of the Danube, including Scandinavia but not Russia, and on the other hand, the lands to the west and south, including the Balkans, the Latin lands, and Great Britain. Production curves of the metal mines in the first part, geologic types and sizes considered, tend to show lower gradients and less variation from an average than those in the other part. In the first part there are many more production curves that continue to rise to the present day than in the second part. Now, in the first part, the Germanic and Scandinavian lands, there are more survivals of State ownership and control than there are in the second part, the Balkan and Latin lands and Great Britain.

I have come to the conclusion that there is a relation between orderly development and State control. Perhaps I should say here that it is not difficult to find objections to State ownership. My visits to Almaden, Joachimsthal and Pribram left me convinced that technique is apt to suffer by State control, but I am also convinced that such control yields impressive benefits.

No thoughtful person can consider the production curves of most of our own outstanding precious-metal districts—Comstock, Goldfield, Tonopah, Leadville, Aspen, Cripple Creek, and many more—90 per cent. of whose output has been gathered within 20 years or less, without wondering whether they have been exploited to the best advantage for the country.

In the matter of absentee ownership, I shall only mention the fact that many of the outstanding metal mines of Spain and Italy have been exploited by British and French capital. It is probably more than a coincidence that in many mines such ownership was acquired after British sources began to fail. In these places, therefore, production curves tend to resemble the British rather than the Germanic types.

Stages in Metal Production

Having discussed thus far only the characteristic features of district production curves of the common metals, I shall take up several aspects of the metals problem viewed more broadly.

I have frequently wondered whether there is any way by which a nation can ascertain the stage of rise or decline of metal production in which it finds itself. We have long known that normal human life is distinguished by several epochs, which are not sharply separable, however—childhood, adolescence, maturity, old age. In recent years we have learned enough about the physical changes in the body so that a physician by examination of the eyes, teeth, blood pressure, etc., only parts of the body, can form a trustworthy opinion of the epoch of life which the whole individual has reached. So I have searched the record for evidence of the stages of life of the metal industries.

The review that I have been able to make suggests the diagram shown in Fig. 7. This diagram indicates five stages, shown by successive culminations of (1) the quantity of exports of crude ore, (2) the number of mines in operation, (3) the number of smelters or refining units in operation, (4) the production of metal from domestic ore, and (5) the quantity of imports of crude ore. In suggesting these five successive stages in the life of the metal industries, I do not insist that the order is invariable over a wide range of history or over the entire world, for fortuitous discovery and sundry acts of governments can slightly change the order, temporarily. I merely urge that this is the normal order of the past in most regions with which I am familiar. It is well shown in most of the European countries with domestic fuel supplies. No significance is to be attached to the relative heights attained by these curves; I wish only to emphasize the successive relation of the peaks.

Take, for instance, the copper, tin and iron industries of England. Copper ore was first encountered in mining tin ore, and cast aside; there is no record of exports. Then follow peaks of number of mines operated and production of metal from domestic ore (1861). Ore was first imported in 1826, attained a peak in 1889, and has rather steadily declined since. Domestic production has been negligible since 1895, and the Swansea smelters are sustained on imported ore and matte. I have not found it recorded whether tin ore or only the metal was first exported from England in the days of Roman occupation. In the last 200 years we see the rise of domestic production to the peak of 1871. The import trade in tin ore began about 1850, and in 1913 ore imports were four times domestic production. The mining and smelting of iron ore in England also goes back to Roman days. The peak of production of domestic ore was passed in 1882, but imports began about 1865. In recent years domestic production has steadily declined and imports have almost as steadily increased, so that now the iron recovered from the two sources is about equal.

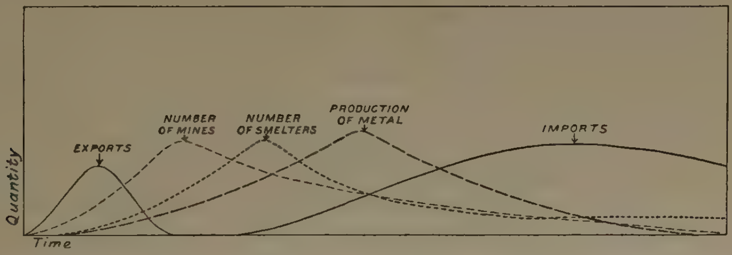

FIG. 7.—STAGES OF THE METAL INDUSTRIES FOR COUNTRIES WITH FUELS.

In Belgium the production of zinc began in 1837, but ore was probably exported earlier to Germany, as it was the early seat of zinc smelting in Europe. Production of domestic ore passed the peak in the decade 1851–1860 and has since ceased. The imports and the production of metal from imported ore have risen steadily to the present day. Similarly, the production of lead ore and iron ore passed the peaks between 1860 and 1870, but gave rise to smelting industries which treat increasing quantities of imported ore. The zinc, lead and iron industries of Germany show clearly the same tendencies.

In our own country, it is interesting to recall that the first ore from many copper, zinc and manganese districts, as well as our first crude iron, were shipped to Europe for treatment, but the period of export of our crude minerals has been uniformly brief. In the study of a number of metal-producing districts in the United States I have noted the same tendency; first, a peak in number of mines operating, next a peak in numbers of smelting or refining units, and then peaks in production of the metals. That the peaks of production of the metals from many of the principal districts, especially in our Rocky Mountain and Sierra States, has been passed as long as present price levels continue I firmly believe. To what extent, as a nation, we have passed or are passing the peaks of production for the metals, will not be discussed because more elements enter into such forecasts than I wish to raise here.

The picture of national metal production that I am sketching is obviously drawn by empirical methods. Is it possible that this picture rests upon so sound a basis as to assume the aspect of a rule? I am inclined to believe that it rests upon sound geologic and technical experience. In the early stages of exploration of a district or a nation, it is fortuitous and uncommon that the richest and largest bodies are found first. Much exploration must be done and many trial mines must be developed before such bodies are found. Owing to the nature of geologic processes, for every large and profitable ore deposit there are tens, if not hundreds, of small and unprofitable deposits. Also, no matter what may be the state of advance in the arts of mining and metallurgy when a district is found, their local application does not reach the most efficient stage until it is known where the largest and richest bodies of the district lie and what is their mineralogic character. It follows, therefore, that the peak of production tends to come after the peak of number of mines. Finally, with the failure of local sources of ore and the maximum advance in technique, smelters and refiners not only need but attract ore from remote sources, and trade in imported ore naturally begins. Whether and how long metallurgical industries survive depends upon the local fuel and power supplies and the maintenance of competitive technique.

SUCCESSIVE EPOCHS OF METAL PRODUCTION

Historians have long recognized the fact that the outstanding sources of the precious metals since the time of earliest records have generally lain on the frontiers of the existing civilizations. Perhaps that is the reason why metals and minerals have provoked conquests. Only recently has De Launay, in Le Conquête Minérale, noted that as civilization sweeps over its frontiers the new lands tend to yield a series of the metals in succession. To quote:[37]

"If we visualize any one of these new countries whatever, we see it, on the average, realize successively an age of gold, then of silver, then of copper, then of lead, then of zinc, then of iron."

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[37] L. DeLaunay: Le Conquête Minérale, 27. Paris, 1908. Flammarion.

In Fig. 8 this order is presented graphically. De Launay does not point out that not only does this order coincide with that of the present prices offered for the metals, but that this order of prices has been maintained for several hundred years. It represents roughly, also, the order of abundance of the metals as shown by production in recent years. Furthermore, the order tends to apply to some countries in Europe as well as to new lands. For instance, in Great Britain, the peaks of domestic production have been successively, copper, 1861; lead, 1870: tin, 1871; iron ore, 1882. For Spain the order has been copper, 1912; lead, 1888; iron ore, 1908. For Germany before 1914 the production of none of the metals under consideration seems to have attained the peak, but in this inquiry it has been impossible to discriminate confidently between metals of domestic origin and those recovered from imported ores. In the United States we seem to have passed the peak of gold and silver production in 1915, and although the production thus far of copper was highest in 1916, of zinc in 1919, of lead in 1926, and of iron ore in 1917, I would hesitate to say that these years will mark the peak for all time. Perhaps at this point, I should again qualify what has been stated about the successive epochs of metals for any country by saying that the order seems to hold only for the price ranges that have existed in recent years. Obviously, if the relative prices were to change greatly or if the general range of metal prices were to increase with respect to those of other commodities, such as the products of agriculture, great changes in the order would follow.

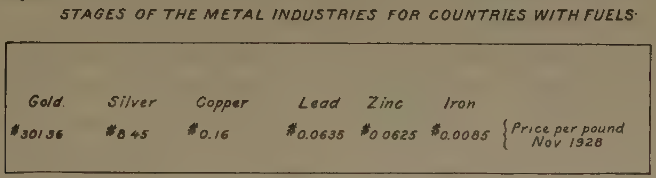

Fig. 8.—DeLaunay's order of exploitation of metals.

I dwell on this succession in the epochs of the metals because it seems to me to reveal several basic influences that surround their production. It seems to depend in part upon the relative abundance or availability of the metals for recovery, in part upon man's needs for them as civilization is now constituted, and in part upon the cost of transportation to consuming centers. Viewed abstractly, it should be possible for a large nation, having entered the iron age, to regulate prices so as to return to the epoch of gold or silver for a period, but the chance that any nation will do so seems very remote. In contemplating the future for the United States we shall probably be wise if we assume that our gold and silver epochs are past.

The immediate outlook for an increase in metal production in Europe is not encouraging, iron possibly being excepted. Whether her statesmen would admit it or not, much of European diplomacy seems to reflect concern over a downward tendency of production and attempts to extend control over it. It is small wonder that continental Europe is turning rapidly toward cartels and understandings—first, iron and steel, next mercury, aluminum and zinc, with lead and copper in the near distance.

STATUS OF THE UNITED STATES

It is not the purpose of this inquiry to consider all or even most of the elements that deserve consideration in any attempt to make a confident forecast of the near or remote outlook for metal production in the United States. The behavior under exploitation of the outstanding sources of the metals in Europe has been reviewed in order that methods may be devised by which our own resources may be studied. On the charts for iron ore, copper, lead, silver and mercury, graphs of outstanding American districts are presented for comparison with those of Europe. From this inquiry I think that the graphs of district production reflect not only the behavior of many types of ore deposits exploited under advancing technique and declining price, but indicate also what may be expected from them during the next few decades, as well as that more subtle influence the national point of view toward mineral exploitation. The remote outlook for a mining district of course can not be determined soundly without penetrating insight into the basic geologic factors as well as other influences that will surround production.

Only a casual examination of the district production graphs and the number of sources is needed to show how marvelously America has been endowed with metals. In number, size, and grade our resources once greatly exceeded or still exceed those of Europe in iron, copper, lead, zinc, silver and gold. Beginning with Europe's store of science and technique, we have attacked those resources vigorously under a system based largely on individual initiative and uncontrolled personal profit. Measured by district production curves, many of our outstanding districts of a few years ago show signs of exhaustion, and we are sustaining or increasing national totals by turning quickly to new sources of lower grade. If science and technique can keep up the pace, we have no cause for concern. Personally, I doubt that they can much longer.

It is worth while to speculate concerning the place of metal production in a modern industrial civilization such as ours. Is the production of metals largely a process for creating capital through profits or is it to be regarded more as one of many industries of the nation where national concern warrants coordination? Further, one well may ask whether a nation should not be dominated by one motive at one stage of its life and another motive at another stage?

For my part, I think that as long as there are frontiers or unexplored regions in a nation, there are outstanding reasons why incentive should be offered toward creating capital through individual profit. At a later stage, when exploration has shown the location, extent and grade of the nation's resources it is worth debating whether such controls should not be placed in effect as will insure such coordination of industries as will aid the orderly advance of production by all industries.

The review that has been made seems to show that in central and northern Europe, certain kinds of control in production of metals have led to a more orderly advance and longer life than in other parts where such controls are largely lacking. It is my belief also that in central and northern Europe metal production is better coordinated with other industries and with other expressions of national life.

To us, a democracy, controls of any sort are obnoxious. Such as we possess have developed slowly and are still resented by many among us. Most of the controls in effect in Europe arise out of state ownership and tend to show the minor evils of bureaucracies. Perhaps, as an employee of our Federal Government, I should say that I do not favor governmental control. We shall make a splendid contribution to democratic government if we can perfect that control of metal production by the industries which shall assure orderly production over long life with a reasonable profit to the industry and proper guarantees to the consuming public.

ACKNOWLEDGMENT

The writer cordially acknowledges the editorial criticism of his paper by B. H. Lane.

DISCUSSION

F. L. GARRISON, Philadelphia, Pa. (written discussion).—The paper by D. F. Hewett cannot fail to revive some of the serious reflections in the minds of those who are becoming progressively concerned by the depletion of the metallic mineral resources of this country. The time has passed when one may complacently dismiss the subject with a hope that any anxiety of this kind for the future will be measurably dispelled in some way—perhaps by some new metals or some physical and chemical process. Already this hope has been realized to an appreciable extent. Improved methods of concentration, metallurgical treatment and mechanical apparatus for mass extraction and transportation have made possible the mining, milling, and utilization of ores previously considered too lean to be of commercial value. Advances in chemistry and metallurgy have placed in abundant use at least one metal which 40 years ago was little else than a laboratory product. I refer to aluminum, for which our then large deposits of bauxite seemed to assure an unlimited supply of raw material. Yet today a large proportion of the metal made in the United States is derived from imported bauxite. Hence it would seem that if it is not going to be possible to make this metal from aluminum silicates, such as kyanite, sillimanite and andalusite, of which we have in the United States unlimited quantities, our production of aluminum from domestic ore resources may be considerably restricted. Fortunately, I understand, experimental methods with the object of producing aluminum from its silicates have yielded promising results and, if ultimately successful, will be a striking example of the creative power of research that is probably the most hopeful augury for the future.

When, 75 years ago, J. D. Whitney wrote his classic on the mineral wealth of the United States,[38] the mineral resources of the country were assumed to be practically unlimited. Whitney's investigations were confined to districts east of the Rocky Mountains. Very little was known at that time of the vast areas to the west which, with the exception of California, did not begin to produce any appreciable amount of metallic minerals until about the time of the Civil War.

In order better to understand the whole subject of mineral depletion as the question now confronts us, it may be desirable at this point to mention briefly some of the salient factors that profoundly controlled and affected the national origin and development of the United States. Unlike the Spanish colonizations in Mexico, Central and South America, which antedated the English and Dutch settlements along the Atlantic seaboard to the north, our first colonists were impelled in seeking new homes by a desire for religious and political freedom with the hope of ultimate better living conditions. As is well known, this was achieved in the face of active, barbarous French and Indian opposition, which would have baffled any but a strong, determined people possessing what is now called "a frontier spirit." Whatever may have been the predominating mercenary motives of the earliest English settlers in Virginia, the sturdy people who later made their homes here and all along the Atlantic coast certainly had no golden objective like the impelling force that was behind the Spanish settlements, or the fur trade, which initiated the French entry into Canada. The English colonists knew nothing of mining and it was not until early in the eighteenth century that they began to smelt the iron ores found in numerous places near their homes. The Colonial industries in the North were agriculture, fishing, whaling and maritime trade, while the South devoted itself at first mainly to the growth of tobacco and later to the cultivation of cotton with slave labor. Thus were established two different systems of industry and culture from which eventually arose an armed conflict that nearly destroyed the United States.

Early Mining in the United States

In 1709, the first chartered mining company in the colonies came into existence for the purpose of working some copper deposits at Simsbury, Conn.[39] Similar copper minerals were subsequently discovered at the Schuyler mine in New Jersey in 1719, and a considerable amount of this ore was shipped to England for treatment. Although the French were aware as early as 1720 of the existence of lead deposits in the Mississippi Valley, no serious or intelligent attempts were made in the way of development until the latter part of the century, when Dubuque commenced mining operations near the site of the present town bearing his name. Hence until this vast region passed into the possession of the United States by the "Louisiana Purchase," its mineral production appears to have been negligible.

During 1829 and 1830, the production of gold from the alluvials in several of the southern states was considerable. Whitney stated that in 1833 and 1834, the amount thus obtained from Virginia, North Carolina, South Carolina and Georgia was about $1,000,000 per year; later the amount fell off to half that sum, then rose again when mines in solid rock were begun.[40] Much if not most of the labor thus employed was slave in the winter season, when it was not otherwise occupied. This gold-mining industry was considered at that time of sufficient importance to cause the government to establish a mint at Charlotte, N. C., now long abandoned because of the decline of the gold-mining industry in the South.

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[38] J. D. Whitney: The Metallic Wealth of the United States. Philadelphia, 1854. Lippincott. London, 1854. Trübner & Co.

[39] *Idem*, 23.

[40] *Idem*, 24.

The lead regions of the Mississippi Valley produced largely during the same period as the gold production in the South, but the nature of the ores was such that systematic operations in this, as in practically all other mining sections of the United States, were not carried on with sufficient engineering and technical skill to ensure reasonable success in attracting the capital requisite for profitable working.

The existence of metallic copper on the south shore of Lake Superior was known to the French Jesuit missionaries as early as the year 1669. Its occurrence is described at length in Father Dablon's "Relation" for 1669–70.[41] As early as 1717, the natives trading at the Hudson Bay Co. fort at Nelson exhibited weapons of copper and spoke of the banks of a distant river in the far north where copper could be picked up freely. This copper probably came from what is now known as the Coppermine River, which flows north into the Arctic through Coronation Gulf. It was not until 1844, however, that the Lake Superior copper deposits became well enough known to attract sufficient attention and capital to begin their development. Five years later the unprecedented gold discoveries in California gave an impetus to all kinds of speculation relating to mining, good, bad and indifferent, but with the valuable practical result of making mining an established industry of the United States. Closely following this period, or about the time of the Civil War, came the extraordinary discoveries of silver-lead ores in the Comstock Lode and at Eureka, Nev.; then those of Georgetown and Boulder company, Colorado, followed; about the same time, the gold-copper and alluvial gold deposits in Montana were discovered. The cumulative effect of these developments was to electrify the country: foreign miners and metallurgists poured in to the West, the technical development of our American mining industry was fairly started on its career, and the United States became the largest producer of precious metals in the world.

At the beginning of the Civil War the iron industry of the northern states was much behind that of England and in the South may be said to have hardly existed. Thus, for example, when the Federal Government undertook to build an ironclad vessel (Monitor) to meet the menace of a similar one being constructed by the Confederate Government, the turret armor had to be made up of eight 1-in. iron plates bolted together because there was not a mill in the country able to roll anything thicker, whereas in England, 4½-in. solid iron tongued-and-grooved plates[42] were placed on one of the new warships built in 1861. Anyone familiar with ballistics knows that eight 1-in. superposed iron plates would be an inferior defence to a 4½-in. solid plate. I might add it was not until 1891 that the United States seriously undertook the manufacture of modern steel armorplate.

DEVELOPMENT IN SEVENTY-FIVE YEARS

The men who were lured to California in the "gold rush of '49," with probably few exceptions, had had no mining experience and their operations were confined to washing gold from the gravels and sands of the Sierra Nevada mountain valleys and gulches. Such work can hardly be dignified as mining, but their activities subsequently developed into great hydraulic washings and lode, or so-called quartz mining.

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[41] See J. W. Foster and J. D. Whitney: Report on the Geology and Topography of a Portion of the Lake Superior Land District, pt. I, 10. *Executive Document* 69. Washington, 1850.

[42] F. L. Garrison: The Development of American Armor Plate. *Jnl.* Franklin Inst. (1892) **133**, 341.

It would therefore seem that our American mining industry of today has developed in the short span of not more than 75 years, yet it would now appear that we are faced with the proximate exhaustion of our metallic mineral reserves and the certainty hereafter of being obliged to depend on the lean ores which remain and the importation of better grades from Alaska and foreign sources. The truth of this situation appears to be realized to some extent by mining engineers if not also by some of the nontechnical men engaged in the industry. It is claimed in some quarters that our accumulated stock of metals is so large and present methods of scrap recovery so efficient that the pinch of scarcity is not likely to be felt for a long time. While this short view no doubt has some justification, it must be remembered that no small proportion of all the metals used in industry are lost or consumed in it beyond recovery. Such a plea seems to me specious, misleading and likely to postpone efforts to establish a sensible, comprehensive national policy of conservation. The United States is no longer, if in fact it ever was, a self-sufficient, self-sustained country. The World War proved that we were deficient in a number of essential metals such as tin, chrome, platinum, tungsten, mercury and nickel. In an emergency we may depend on our Canadian neighbors for nickel; but of the others we have either very little or practically none at all. Of tin, the most important and indispensable in this group, we possess small quantities at widely scattered localities, none of which has yet proved to be of sufficient importance to justify the necessary capital investment for exploitation. As far as I know, there is no tin in Canada and little or none in Mexico. Perhaps it is not generally realized that today most of the iron ore used by the steel works along the Atlantic coast is imported from various parts of the world.

Mineral Deposits Wasting Assets

It is difficult for the public to understand that a mine is a wasting asset and whatever comes out of it is never replaced. Ores do not grow in the ground as crops and trees do on the surface, yet to us this self-evident fact does not always appear to be realized by the public. Moreover, one cannot foretell, save more or less approximately, how much mineral a mine contains until it has been worked out and every nook and cranny of its orebodies explored. It is true that new orebodies are often found in old, sometimes supposedly exhausted mines and that is one of the reserves we will have to depend on in the future. There are today in the United States, as in most other countries, many more abandoned or shut-down mines than going and producing operations. If any virgin metal-bearing areas of notable importance have been discovered recently in this country, I have not heard of them, and the need for their finding has never been greater.

Geophysical prospecting, which is now coming into prominence as a practical aid in mineral discovery and development, is of unquestionable value in locating deposits covered with glacial drift and the "wash" which fills the valleys and gulleys of the semi-arid districts of our western states. This form of prospecting probably has distinct limitations under other conditions, and too much should not be expected of it as a science not yet fully developed. Moreover, our mining laws relating to the public domain are not yet attuned to this form of prospecting and development, hence some changes must be made in them to accommodate locations where no outcrops occur.

Perhaps it is not going too far to say that today the cream of our metallic mineral deposits has been taken off, and to make profitable the skimmed milk of what is left will require all the resources of modern science. Statistics of production and peaks in plotted graphs may be misleading, since they indicate something produced by processes that are assumed to continue indefinitely upon a basis of a natural raw material supply without limitations. The production lost from exhausted or inoperative mines is now being supplied by increased volume of output at places wheer various mechanical, chemical and metallurgical methods have enabled operators to use leaner, complex, and refractory ores previously not amenable to such systems. This is well illustrated at our so-called "porphyry" copper mines and others where extraction has attained a high degree of efficiency and the loss in tailings is reduced to a minimum.

Mr. Hewett's opinion is that in the earlier stages of the exploration of a mineral district it is an uncommon experience for the largest and richest orebodies to be the first discovered. If this is considered a rule, it has had so many exceptions, once the discovery and early prospecting stage is passed, that one might be disposed to believe general experience had been just about the contrary. In other words, the best orebodies more frequently are those that outcrop and are the first to be discovered. This thought is important, because we may go on believing that present conditions are not as bad as they appear and the largest and best orebodies are yet to be found. In many if not most instances, the better ores are near the surface and are usually skimmed off before the mine settles down to steady and uniform production, mostly on ores of lower grade than those first found.

Since the dawn of recorded history man has used metals of some kind and in some manner, but so recently as a century ago the amount of metal available to him was relatively small as compared with the enormous quantities now in use. Hence when we find metal mines in Europe 300 or 400 years old and still producing, we are not justified in concluding they originally were, or are now, any better as a whole than are those in this country. Probably under similar operating conditions and care, our mines would last as long, for the longevity of those in Europe is due to low rate of production with economical operation; moreover it is more than likely they were nursed along by the governments and rulers who had whole or partial ownership of them. Such were the mines Agricola wrote about in the latter part of the sixteenth century, and it was in the neighborhood of these mines that the justly famous German mining academies of Freiberg and Clausthal had their origin.

Governmental Regulation or Ownership

In England no system of governmental or royal ownership exists or appears ever to have existed. A tax ("coinage") on tin after smelting was for centuries paid to the earls and dukes of Cornwall, but this form of royalty or tax was abolished in 1838. Cornwall is a classic country for the mining engineer and although today its mines are assumed to be practically exhausted, it has been claimed by those in a position to know that, were the existing old system of restrictions, customs and land tenure which has grown up under centuries of private and corporate ownership abolished, many of these now idle mines could be operated at a satisfactory profit.

Evidently governmental regulation or ownership of mines is as distasteful to our British cousins as it is to us. Nevertheless is seems likely the time may soon be at hand when something must be done to regulate our mineral industry and check waste of its basic resources, which, unlike our forests, cannot be renewed: especially as we are not likely to find satisfactory substitutes for metals. This same reasoning would apply equally well to Canada, where American capital is becoming so largely engaged. Our northern neighbors have been as wasteful as ourselves and their mining industry has won its enviable place only after a long struggle against hard physical conditions, and lack of previous mining experience and capital. Their mining laws are generally better than ours for they have had the advantage of our experience and the advice of some of our best mining engineers and geologists. The Dominion Geological Survey and most of the provincial geological and mining departments have rendered the greatest possible aid in the sensible direction of the industry throughout Canada.

South of us in Mexico, which is a country possessing great mineral wealth, mines have been operated since the Spanish conquest in 1519–20. As a consequence of several centuries of training and experience Mexican miners are generally good; the climate is healthful and the mining industry would be prosperous were the government more stable and life and property better protected.

If we ever become involved in a serious war and have our coasts blockaded and maritime commerce raided, we may have to depend upon our neighbors to the north and the south for many things we do not have within our own borders. So aside from every other reason their prosperity and development is of vital concern to our own country.

Necessity for Conservation

It has been assumed by some industrialists that the best way to make money is to control natural resources. Chemists claim that a better way is to develop new chemical processes, but even admitting this thesis chemical progress must be built up upon some kind of raw, natural material. With sufficient power or energy it may be possible for some future alchemist to so juggle the atoms that the transmutation of one metal into another will become a commonplace affair. Meanwhile we would do well to conserve what we have, and not trust to future scientific ledgerdemain to make amends for our present lack of prevision and consideration for generations to come.

On the broad question of the conservation of natural resources, agricultural, mineral and fisheries, the three great basic and fundamental means by which we directly draw sustenance from the earth, it would seem that as a nation we have not yet given the subject the attention and serious consideration it evidently deserves. Soil impoverishment, forest waste and the neglect of fish culture and protection are subjects not difficult for the public to visualize and understand. Thus for a simple and relatively unimportant example, only a few years ago the large tidal rivers flowing into the Atlantic on our eastern seaboard teemed in the spring of the year with a delicious fish known as shad. Now this source of food supply has been greatly diminished in most of these rivers, and in others, as our Delaware, wholly destroyed by sewage and oil refuse discharged into the river. Everybody knows about forest wastes and the increasing cost of lumber through loss by fire and careless lumbering. Soil exhaustion by erosion and crop cultivation is less obvious, and mineral depletion is almost wholly unrealized or understood by the general public. Evidently it is only through education we can hope to obtain the necessary public support to inaugurate and sustain a sound conservation policy for dealing with these questions of vital national importance. In regard to forest protection much has already been accomplished by state and Federal governments, which have set aside large areas for that purpose even to the extent, in some of the western states, of reserving for forests large tracts of public lands without a thing on them which might be dignified a tree.

Individualists though we Americans are by tradition and disposition, it is evident that these great conservation questions must be developed by collective action and regulated by legal enactment. Our Federal mining laws today do not fit in with modern conditions, and in view of our present knowledge of economic geology some of them are quite archaic. The question of government ownership will inevitably arise when these subjects are under discussion and it will be said that if the Federal and state governments have taken over and own outright both the timber and the land on which it grows, why should they not also possess the minerals thereon? It is true our local and general governments have usually encouraged and aided the mining industry, the Federal authorities especially have been of enormous benefit through the Geological Survey and the Bureau of Mines, but these organizations as well as the individual state authorities could do much more were they backed by an enlightened and sympathetic public.

Air Conditioning in Deep Mines

By R. W. Waterfill,* Newark, N. J.

(New York Meeting, February, 1929)

Many existing ore deposits of valuable metals have been worked out in their upper surface levels and the continued productivity of these mines is dependent on their extension to greater depths in the earth. The maximum depths at present reached are about 6000 to 8000 ft. below the surface, with no evidence of an impoverished ore vein. These mines are widely distributed; some of the best known are in South Africa, Brazil, India and in the western sections of the United States.

Among the factors which increase the difficulty of working these mines at increasing depths are hoisting, rock pressure and temperature. The manner and effectiveness with which these problems are met will, no doubt, determine in a large measure the ultimate depths to which any mine can be profitably worked. The most critical of these three factors at present is the increase in temperature with depth. The high temperature and humidity in many of these mines at present are severely detrimental to the health of the workers and greatly hamper their efficiency. Some have almost reached a condition in which the human system cannot survive. The problems involved in the cooling and ventilation of mines have all been ably analyzed and presented in previous articles. For this reason, only a résumé of conditions affecting the cooling and conditioning of deep and hot mines will be given in this paper before attention is directed to possible methods of improving atmospheric conditions in the working levels.

PHYSIOLOGICAL EFFECTS OF TEMPERATURE, HUMIDITY AND AIR MOTION

Many data on the physiological effects of different atmospheric conditions on the human system have been compiled and published through the combined efforts of the American Society of Heating and Ventilating Engineers, the U. S. Bureau of Mines and U. S. Public Health Service. These data were not primarily developed around the conditions existing in deep mines, but the upper temperature limits of the investigations extend into, and are applicable to, mine atmospheric conditions. While the exact values of the standard comfort conditions, established for persons of such habits as those employed in detecting the different degrees of comfort, might not produce identical effects upon persons inured to the conditions existing in mines, it is not believed that the variation from this standard is great.

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* Carrier Engineering Corporation.

Body temperatures will vary slightly with external atmospheric conditions in an endeavor to dissipate body heat, but the permissible variation is not great, nor can we say there is a very wide difference in human races. It is true some peoples are more resistant to high temperatures, others to lower temperatures, but there are general conditions under which all appear to be more healthy, productive and efficient. The

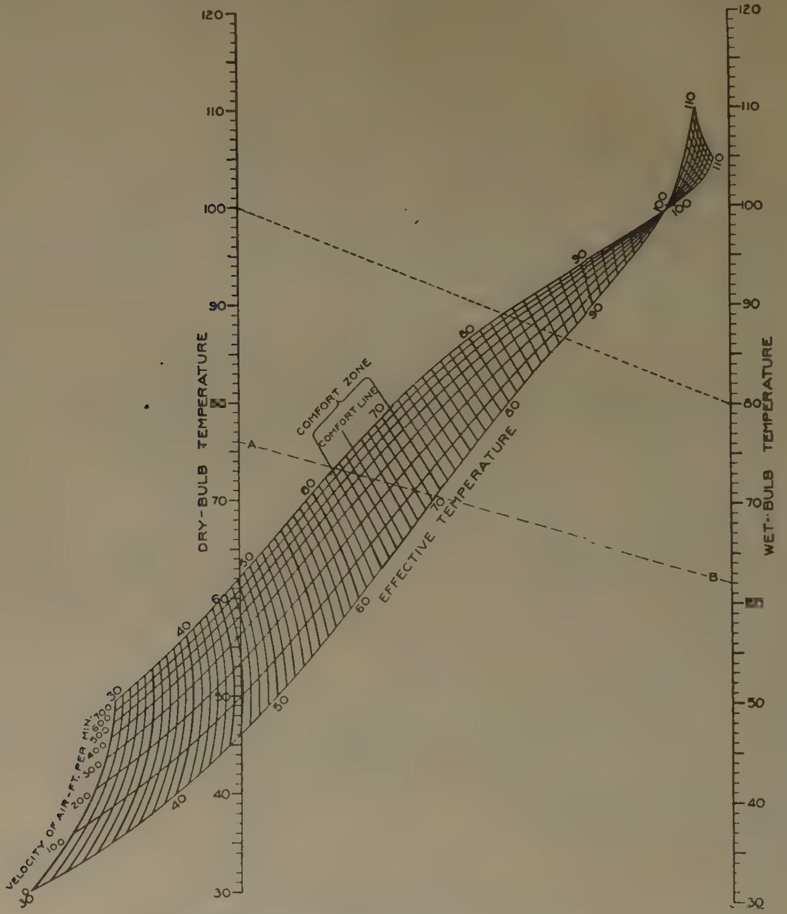

FIG. 1.—EFFECTIVE TEMPERATURE CHART.

For determining "sensible" or effective temperature for individuals normally clothed, from measurements of dry-bulb temperature, wet-bulb temperature and velocity of air. ("Effective temperature" may be defined as "the combination of dry-bulb temperature, relative humidity and air motion which produces a given comfort effect on human beings; in other words, it is the result of using the human body as a temperature-measuring device.")

Chart compiled and published by the American Society of Heating and Ventilating Engineers in cooperation with the U. S. Bureau of Mines.

results obtained at the U. S. Bureau of Mines at Pittsburgh, Pa., indicate an ideal condition sufficiently accurate for all practical purposes, and what is just as important, the tendencies and relative effects of variations from the ideal. They have established the fundamental relations of waste-heat dissipation from the human machine and conditions of comfort or efficiency.

The manner in which the human system disposes of the heat generated by its physical and chemical activities is by radiation, convection and evaporation. It has thus been established that there are certain relations between body and skin temperatures and the temperature, humidity and velocity of the surrounding air. These relations have been recorded graphically in Fig. 1. This chart shows the relations of dry-bulb and wet-bulb temperatures and of air motion to effective temperatures, a given effective temperature including all the combinations of those three factors which produce the same degree of comfort. It will be noted that in the higher temperature range, that is, at temperatures normally existing in some mines, the cooling effect of air velocity is small, and the effective temperature approaches the wet-bulb temperature. At about 100° F. effective temperature, there is no difference in cooling effect between still air and air in rapid circulation. At higher effective temperatures, air circulation becomes detrimental rather than beneficial as a cooling means.

At 100° F. dry bulb and 80° F. wet bulb, the effective temperature in still air is 86.5° F. By increasing the air velocity to 700 ft. per min., the effective temperature can be reduced to 83.5° F. This gives a reduction in effective temperature of 3° F. by considerably increasing the air velocity. Very little gain, however, would result from boosting this velocity to 1000 ft. per min. By extrapolation it is found that this velocity reduces the effective temperature another half degree.

Our conclusions from a study of this chart would be that in some deep mines there is little to be gained in cooling effect by simply increasing the velocity. This increase, of course, may mean a greater volume of air, which might lower the air temperature in the mine. The economic balance of the greater power required for increased circulation, as against the net gain in cooling effect, offers interesting material for study and analysis in itself.

At the higher temperatures, the loss of heat by radiation and convection is small. Most of the heat is removed by evaporation and consequently the wet-bulb temperature of the air plays a very important part in its cooling effect.

Another interesting fact established at the Bureau of Mines is the effect of clothing on comfort at the higher temperature.[1] In still air with 94° F. effective temperature, it made no difference whether the subject was normally clothed or stripped. Below this temperature, human beings were cooler when not clothed, whereas at higher temperature clothing was beneficial in that it aided cooling by a greater evaporation.

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[1] C. P. Yaglou and W. E. Miller: Effective Temperature with Clothing. *Trans.* Amer. Soc. Heating and Ventilating Engrs. (1925) **31**, 89.

These results indicate the effects of atmospheric conditions on health and comfort alone. That health is a necessity to human efficiency is a universally accepted truth. Relative results of the effect of comfort on efficiency have also been investigated.[2]

These tests indicate that with air at 100° F. dry-bulb temperature and 100 per cent. R. H., there was no difference in the amount of work accomplished, whether the air was still or in motion. The work accomplished at this condition was only about 40 per cent. of that at 90° F. and saturated, and 30 per cent. of that at 100° F. dry bulb and 60 per cent. R. H. The fatigue in each case was as nearly equal as could be measured.

From these data it appears that for the atmospheric conditions having an effective temperature approximately the same as body temperature the cooling effects of high-velocity air and of the removal of clothing are nil, and further that the physical activity of those exposed to such conditions necessarily approaches a minimum.

Sources of Heat in Mines

The general sources of heat in the mine are wall surfaces, men, explosives, machinery, and the cooling of new rock as opened up. This latter source is greater than all others combined and increases with depth, or the rock temperature. In most of the deepest metal mines, these sources produce virtually all of the heat load. There are, however, other mines in which heat is evolved by some form of chemical reaction.

The present method of cooling these mines is chiefly by circulation of large quantities of air throughout the workings. This air, however, in passing through the mine from the upper to the lower depth is heated by adiabatic compression and by the increasing rock temperature. It is evident, therefore, that if we establish a maximum permissible temperature at the bottom of the mine, the difference between this temperature and the actual temperature of the air due to adiabatic compression and rock heat sources is gradually diminished, or, in other words, as the depth is increased, the useful cooling effect per pound of air approaches zero. Since the adiabatic and rock temperature effects are due to depth alone, it is further evident that the desired atmospheric conditions cannot be maintained by increased air circulation alone. It has been estimated that this condition generally limits the depth to which mines can be successfully worked, in most regions without some means of cooling the the air, to about 7000 or 8000 feet.

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[2] W. J. McConnell and C. P. Yaglou: Work Tests Conducted in Atmospheres of High Temperatures and Various Humidities in Still and Moving Air. *Trans.* Amer. Soc. Heating and Ventilating Engrs. (1925) **31**, 101.

The work of compression of air or any other gas in passing from one state to another of higher pressure is the same regardless of the means used to accomplish the compression. This, of course, is modified by the understanding that the efficiency of compression is the same. Thus it makes no difference whether the air is compressed mechanically or by the gravitational weight of its own columnar mass, in passing from the surface to the bottom of the mine. If we have an adiabatic column of air of a given height, the barometric pressure of this column is related to the height at any point thus:[3]

$$h_2 - h_1 = \left(\frac{k}{k-1}\right) RT \left[\left(\frac{P_2}{P_1}\right)^{\frac{k-1}{k}} - 1\right]$$

The work of compression when the air from the upper part of the column is caused to pass to the bottom occurs adiabatically. This work may be expressed by the relation

$$W = \frac{P_1V_1 - P_2V_2}{(k-1)}$$

The temperature resulting from this compression is represented by the relation

$$\frac{T_2}{T_1} = \left(\frac{P_2}{P_1}\right)^{\frac{k-1}{k}}$$

h_2 and h_1 = final and initial heights respectively.
R = gas constant.
W = work of compression.
P_1 = initial absolute pressure.
P_2 = final absolute pressure.
T_1 = initial dry-bulb temperature of air, degrees absolute.
T_2 = final dry-bulb temperature of air, degrees absolute.
k = ratio of specific heats, or adiabatic exponent.

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[3] Derived from the relations for work involved, by W. H. Carrier: Centrifugal Compression as Applied to Refrigeration. *Refrig Eng.* (1926) **12,** 253.

An approximate formula given in American Civil Engineers' Handbook (1920, Mansfield Merriman) for establishing differences in elevation from barometric readings is:

$$D = 60158.6\,(\log h - \log H)\left[1 + \frac{t'_a - t_a - 64}{900}\right]$$

D = difference in elevation in feet.
h = height of mercury column at lower station.
H = height of mercury column at higher station.
t'_a and t_a are the observed air temperatures.

Fig. 2 shows the adiabatic dry-bulb temperature gradient of the air from the top to the bottom of a given column. This results only from the heat added to the air due to the change of pressure. The wet-bulb temperatures corresponding to this rate of total heat gain will increase as indicated by the dotted lines.

There are, of course, other sources of heat in a mine. To this natural temperature rise must be added the heat from the rock surface, which in general is in the form of moisture and mainly increases the wet-bulb temperature. Friction further adds a small amount of heat to the air. These effects are to a degree cumulative but tend to combine in such a manner as to approach a saturated condition very close to the existing rock temperature at the different levels.

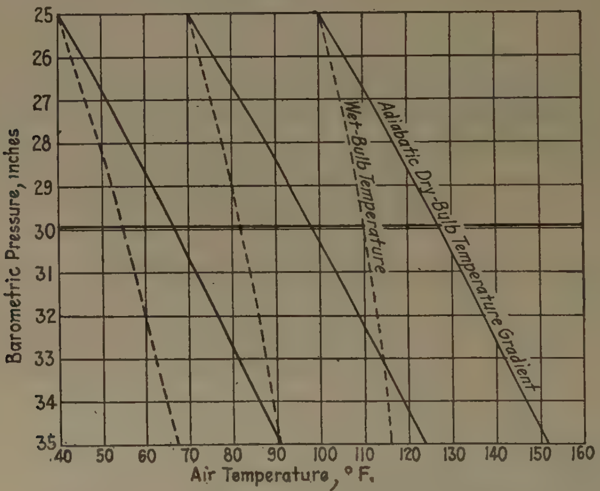

Fig. 2.—Adiabatic temperature gradient.

The analysis of the causes of the existing atmospheric conditions in the lower levels of deep mines shows conclusively and accurately the seriousness of this problem at greater depths. The necessity for cooling is firmly established by experience and by the results of studies of the physiological effects of different psychometric conditions.

Probably the most practical means of cooling is by refrigeration. The desirable location of the cooling and conditioning plant might be questioned. It may be located at the surface outside of the mine or at some point below the surface. Since we are primarily concerned with the cooling of the portion of the mine that is being worked, it is evident that this is the zone in which we want the coldest air. Theoretically it would be possible to cool the air at the surface as well as at the working levels by simply cooling to a lower temperature to allow for the natural rise due to adiabatic compression, but the results of records at one mine, by Tillard and

Ranson[4] showed that there is very little difference in the final wet-bulb temperature of the air after it has passed the 6000-ft. level when the entering conditions at the surface range from 32° wet bulb in winter to 62° wet bulb in summer. The observations of others confirm this tendency. This would indicate that the purpose of a low-temperature air at the surface is largely defeated by the increased heat absorption before reaching the areas to be cooled. J. A. Jones[5] has effectively expressed this result in his conclusions after analyzing the existing influences. First, speaking of the relative amounts of heat to be extracted in surface and subsurface plants, he continues:

> This would appear to indicate the superiority of a surface cooling plant. But consider what happens when the cold air is passed down into a hot mine in this way. The air heats up rapidly, both by adiabatic compression and by heat-flow from the rocks, this latter being enormously increased by the larger difference of temperature between the air and the rocks. As the air warms up, its affinity for water increases, and the air soon becomes damper. The result is that only a fraction of the cooling effect produced on surface survives at the bottom of the mine.
>
> On the other hand, if the cooling is carried out near the place at which the air is to be used, a much larger percentage of it will be usefully available.
>
> A "positional cooling-plant efficiency" might be defined as
>
> $$\frac{\text{decrease in total heat per pound of dry air in the workings}}{\text{amount of heat abstracted by the cooling plant per pound of dry air}}$$
>
> A surface plant might, in the bottom workings of a mine, have a "positional efficiency" of 50 per cent. while a plant working at the bottom of the mine would have a positional efficiency of nearly 100 per cent. So that, to produce the same cooling effect in the workings, the surface plant would have to be twice the size of the underground plant. Of course, against this great advantage of a decrease in the size of the plant must be placed such disadvantages as the questions of pumping cooling water, underground excavations, etc.

This would indicate clearly the operating advantages of a plant located as near as possible to the working level, so that the cooling effect created may be utilized most advantageously.

There are other factors, however, which demand precedent in determining the location of the cooling system. This was realized at the Morro Velho mine of the St. John del Rey Mining Co. Ltd., Brazil. The location of this plant at the surface was the recognition of the importance of factors which could not be disregarded. In the first place, this represented the most desirable type of equipment available at the time this installation was being designed, about 1914. The cooling plant is enormous in size and would have entailed a very considerable expense for underground installation. Another factor is the character of the refrigerant. A mine is not the place in which to install equipment containing noxious vapors. A third serious difficulty was a suitable and sufficient supply of cooling water at the proper temperature for the refrigerant condenser.

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[4] M. O. Tillard and E. C. Ranson: Rock and Air Temperatures in Deep Level Mines. *Jnl. Chem.* (1926) **26**, No. 8, 184.

[5] J. S. Jones: Hygrometry for Deep Mines. *Trans.* Inst. Min. and Met. (1924) **34**, Pt. 1, 3.

It is evident, therefore, that regardless of the favorable operating advantages of a subsurface cooling and conditioning plant, such a plant presents other problems requiring careful consideration.

A complete cooling system to meet these requirements must be reasonably small and safe in operation and should perform two major functions: (1) it must remove the heat from the downcast air, just before it enters the working places; (2) it must find some means of discharging this heat outside the mine, without returning it to any part of the working zone. The system to be described accomplishes this in four steps. The heat is removed from the entering air by cold water, the water thus warmed is recooled by refrigeration, which transfers the heat to another water circuit. The water in this latter circuit finally disposes of the heat to the upcast air, which carries it from the mine.

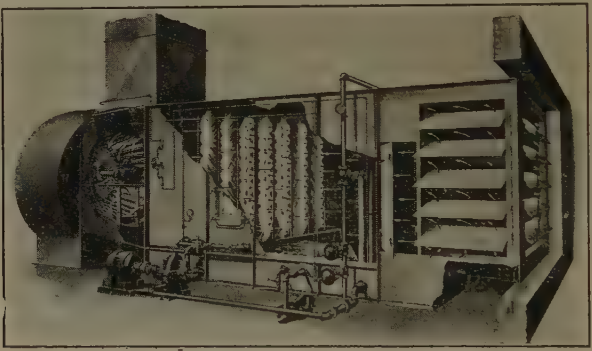

FIG. 3.—SPRAY-TYPE DEHUMIDIFIER.

For cooling and dehumidifying the air, there are a number of methods of heat transfer from the air to the refrigerating medium. The air may be cooled by contact with a cold surface, like a bunker coil, by direct contact with cold spray water, or by a combination of the two, where the air passes through the bunker coils over which water is allowed to shower. The efficiency of heat transfer from the air to bunker coils is such as to require very large surfaces or very low brine or refrigerant temperatures inside of the coil. By directly spraying water into the air, a very large cooling surface, represented by the many individual drops of water spray, is presented, which enables air dew-point temperatures to be obtained, which are very close to the temperature of the cold water leaving the spray chamber. This method of dehumidifying is not applicable to temperatures below 32° F., unless a nonfreezing solution is used.

The cooling of mine air at the working levels, however, does not require such low temperatures. Fig. 3 shows a typical spray-type dehumidifier.

One recently developed refrigerating unit, which has some features that seem favorable to an underground air-conditioning plant, is of the centrifugal compression type. Fig. 4 shows diagrammatically the assembly and operating cycle of the complete system. The compressor is mounted directly on top of the cooler and condenser, which are of standard shell and tube construction. These are the three essential elements and have the same functions in any compression refrigerating system, but vary greatly in form and construction. The motor for driving the compressor is connected through gears to the compressor shaft and mounted on one end of the cooler and condenser. The auxiliaries are a small refrigerant pump and purge pump. The refrigerant pump elevates the refrigerant from the bottom of the cooler to the top and showers is over the tube surface. This gives a flash type of evaporation, which it very efficient. Since this system operates at pressures below atmospheric, a small automatic purge unit for maintaining the system free of air or noncondensable gases is provided.

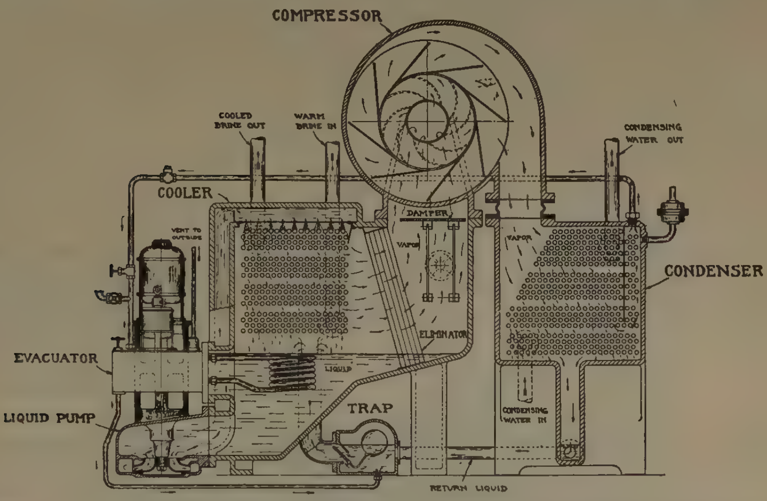

FIG. 4.—DIAGRAMMATIC ARRANGEMENT OF CENTRIFUGAL REFRIGERATION UNIT.

All of these elements are assembled in one compact unit. One of the unique features of this system is the absence of valves. There is no valve in the compressor itself, neither is there an expansion valve nor a valve of any other type in the entire circuit. The only mechanism which might in a sense be regarded as a valve is the float trap, which returns the liquid condensate from the condenser to the cooler. This is not in reality a valve, as it does not require a valve seat; it merely functions as a return trap and is not required to close tightly.

In the centrifugal system, the spray water is pumped directly through the shell and tube-type cooler, Fig. 5, so that the efficiency of heat transfer from the air to the refrigerating medium is high.

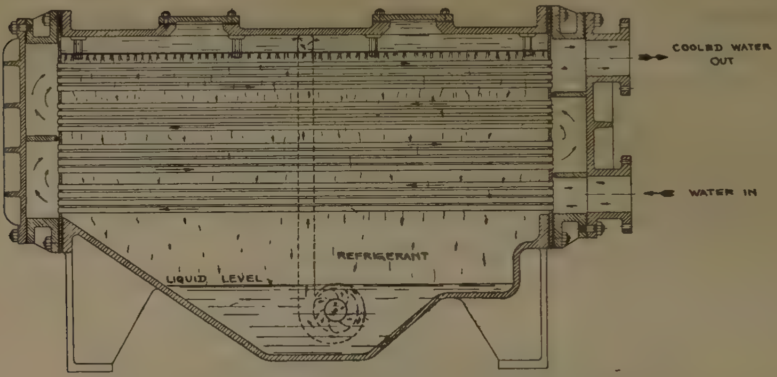

FIG. 5.—CONSTRUCTION OF WATER COOLER IN CENTRIFUGAL REFRIGERATING SYSTEM.

The construction of the compressor is similar to that of a multistage centrifugal pump. Fig. 6 shows the compressor with the top cover removed. The rotor is supported on two standard sleeve bearings which are ring-oiled. There are no other points of contact between moving and stationary parts, which illustrates the simplicity of construction and the absence of sources of mechanical trouble.

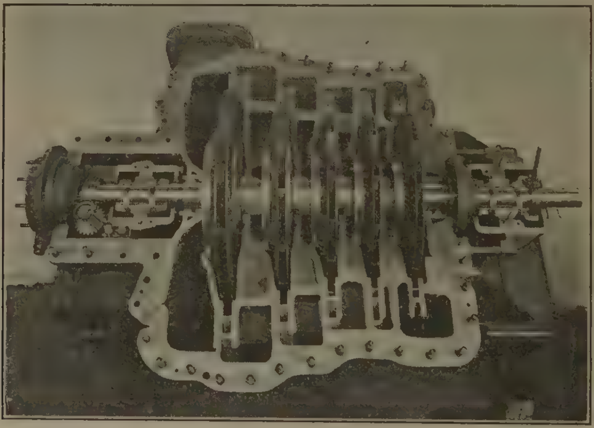

FIG. 6.—INTERIOR VIEW OF CENTRIFUGAL COMPRESSOR.

A centrifugal compressor, like the centrifugal pump or fan, is well suited to accommodate its performance to load requirements. In fact, its operation and control are almost identical with that of the centrifugal fan.

The refrigerant employed in centrifugal systems is unique in the safety and ease with which it may be handled. The most preferred refrigerant is Carrene (CH_2Cl_2). It is not combustible nor does it endanger human life. Being a liquid at normal atmospheric temperatures and pressures, Carrene may be carried in open containers, like water. To charge the refrigerating machine with Carrene, the liquid refrigerant is drawn by vacuum into the system from an open container. Table 1 shows the physical properties of Carrene. Since the normal atmospheric boiling point is 105° F. at 29.92 in. Hg, it is apparent that all parts of the system are at pressures below the surrounding atmosphere at normal operation.

The heat which has been removed from the air by the refrigerating machine must be conveyed by some means from the mine. The upcast air, after it has passed through the working levels, offers the only convenient means for disposal of this heat. A spray system similar to that used for the dehumidifier may be employed for transferring the heat from this warm water to the outgoing air. This will be done at a fairly high temperature level.

TABLE 1.—*Standard-ton Data on Carrene* (CH_2Cl_2)

Temperature	5° F.	86° F.
Absolute pressure Hg., in	2.39	20.5
Vacuum Hg, in	27.53	9.42
Volume-vapor, cu. ft. per lb	49.9	6.7
Density-liquid, cu. ft. per lb	85.6	81.2
Density-vapor, cu. ft. per lb	0.02	0.1494
Heat-liquid, B.t.u. per lb., above 0° F	1.7	29.2
Heat-latent, B.t.u. per lb	162.0	143.5
Heat-vapor, B.t.u. per lb	163.7	172.7
Specific heat of liquid	0.34	
Specific heat of vapor C_p	0.154	
Specific heat of vapor C_v	0.131	
Ratio C_p/C_v	1.18	
Specific gravity-liquid, (water = 1)	1.33	
Specific gravity-vapor, (air = 1)	3.0	
Critical pressure, lb. per sq. in. (Abs.)	1490	
Critical temperature	421° F.	
Molecular weight	84.9	
Melting point (1 Atm.)	−142° F.	
Boiling point (1 Atm.)	105° F.	

By assuming a general hypothetical case the approximate magnitude of the elements involved and the results obtainable may be indicated. Figs. 7, 8 and 9 show such a plant in principle, together with the general scheme of air circulation, for conditioning the air just before it enters the working levels. The dehumidifying apparatus is located about 5000 ft. below sea level and the barometric pressure is 35 in. of mercury. The capacity of the assumed plant is 5600 lb. of air per min. This air may be treated in two ways. In the illustration, only part of the air is passed through the dehumidifier, where it is cooled to a low dew-point temperature and then mixed with the remaining air. The alternate would be to have a larger dehumidifier and cool the total air mass, but to a different dew-point temperature.

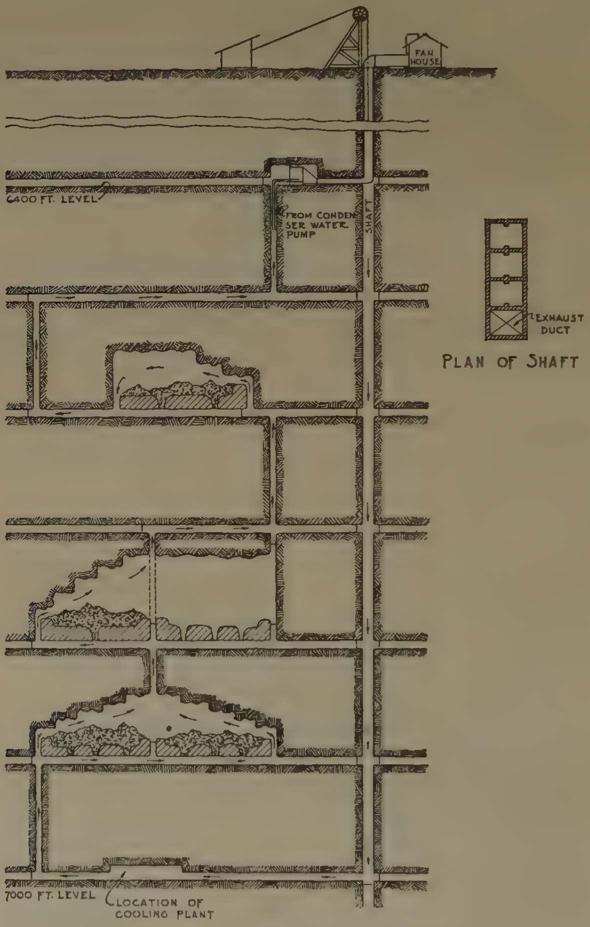

Fig. 7.—Ventilation diagram.

The air-conditioning apparatus room shown is about 24 by 40 ft. by 12 ft. high. Different proportions would be obtained by changing the arrangement. The air enters at a dry-bulb temperature of 100° F. and a wet-bulb temperature of 75° F. Part of this is passed through the dehumidifier, where it is cooled to a low dew-point temperature. This portion of cooled air is mixed with the uncooled portion, producing a resultant air volume at a temperature of 75° dry bulb and 64° wet bulb. The air then enters the working zone and after passing through all of the working levels its temperature rises to a condition of 100° F. dry bulb and 80° F. wet bulb, and it has absorbed in its passage some 80,000 B.t.u. per minute.

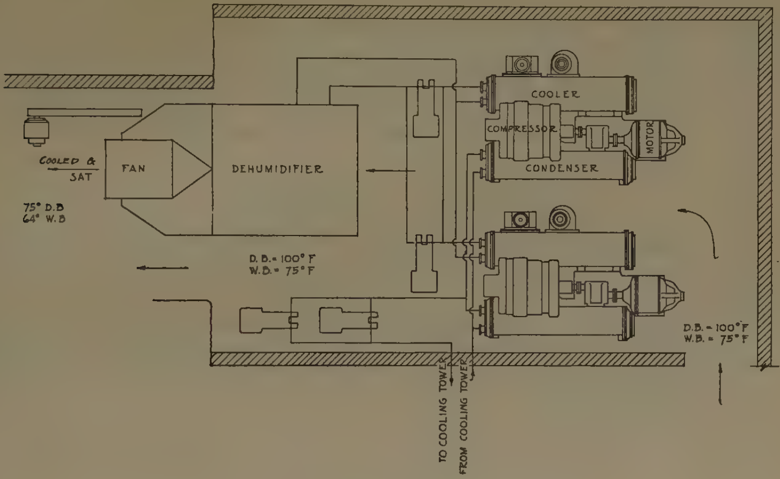

FIG. 8.—AIR-CONDITIONING PLANT.

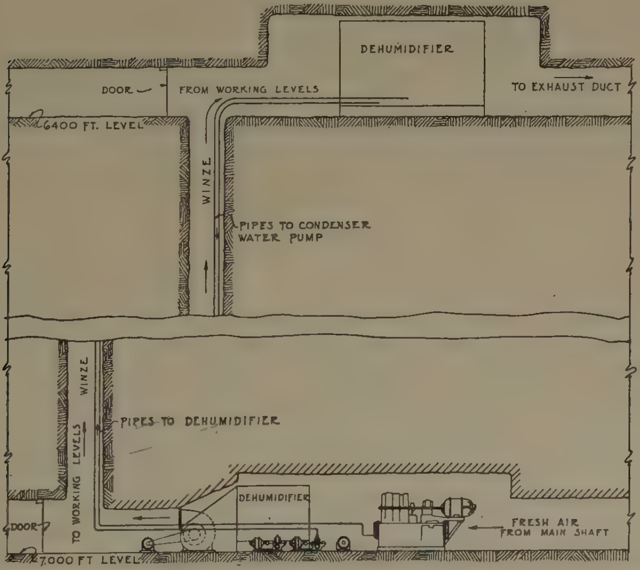

FIG. 9.—ELEVATION AIR CONDITIONING AND CONDENSER WATER-COOLING EQUIPMENT.

The air at this final temperature, before it is removed from the mine, is passed through a humidifier which serves the purpose of a cooling tower. This air is used to cool the condenser cooling water. The water in this case comes from the condenser at a temperature of say 100° F. and after being cooled by partial evaporation in the humidifier leaves at 90° F. to return to the refrigerating equipment for recirculation. The air in passing through this spray water has its own wet-bulb temperature raised by having absorbed heat from the water and leaves the spray chambers at 90° F. saturated. From this point it is exhausted directly from the mine. This illustrates only the general features and possibilities of a subsurface air-conditioning plant. Each individual mine, no doubt, has its specific problems, which would introduce many variations in detail.

ADDITIONAL REFERENCES

1. W. J. McConnell and F. C. Houghten: Some Physiological Reactions to High Temperatures and Humidities. *Jnl.* Amer. Soc. Heating and Ventilating Engrs. (1923) **29**, 131.
2. W. J. McConnell, F. C. Houghten and F. M. Phillips: Further Study of Physiological Reactions. *Jnl.* Amer. Soc. Heating and Ventilating Engrs. (1923) **29**, 507.
3. W. J. McConnell: Effects of High Temperatures and Humidities. *The Nation's Health* (1922) **4**, 616.
4. E. G. Lawford: Commercial Air Conditioning. *Eng. & Min. Jnl.* (1928) **125**, 5.
5. H. L. Callendar: Report on City Deep Mine. 1921.
6. Refrigeration in Mines. *Cold Storage* [London. Ed., J. Raymond] (1922) **25**, No. 294, 266.
7. J. H. Veasey: The Cooling of Mine Air. *Jnl.* Chem., Met. and Min. Soc. of South Africa (1925) **25**, 196.

Note.—The references on comfort conditions and physiological reactions and other papers not mentioned are the results of investigations made through the combined efforts of the American Society of Heating and Ventilating Engineers, the U. S. Bureau of Mines and U. S. Public Health Service.

DISCUSSION

E. Davies, Morro Velho, Brazil (written discussion).—The system of underground air cooling which Mr. Waterfill describes would seem, in theory at any rate, to be the most advantageous that could possibly be conceived from the point of view of the mine operator, because (1) the refrigerant is nonpoisonous and need not be stored under pressure, and (2) a high-speed turbocompressor is employed, which reduces to a minimum the over-all dimensions of the plant and, consequently, of the excavation to contain it, and results in a lower total cost per unit of refrigeration than could be obtained by other means.

The value of the paper would be increased if Mr. Waterfill would give the power consumption of the hypothetical underground plant which he describes and illustrates in Figs. 7, 8 and 9. This power consumption would be, approximately, the sum of the following three items:

(1) Power input to compressor motors.

(2) Power input to the pumps supplying cooled water to the sprays of the air-conditioning dehumidifier.

(3) Power input to the pumps raising the warmed condenser water through the assumed height of 600 ft. to the condenser water cooler and forcing it through the sprays of the latter.

Some of the power put into the condenser water pumps would be obtainable from the cooled water returning from the condenser water cooler, which would reach the main plant at a head of 600 ft., less pipe friction, and therefore could be employed to drive a Pelton wheel or other form of hydraulic motor. Even so, probably the power consumed under item 3 would amount to a considerable percentage of the total requirements. When the comparatively high water pressure involved is considered it will also be realized that the cost of the upgoing and downcoming water pipe lines, together with that of the pumps and hydraulic motors, would form an appreciable proportion of the total outlay for the plant, and for these reasons I am of the opinion that in the case of an underground installation such as Mr. Waterfill describes it might pay to place the main cooling plant at the same level as the condenser water cooler: *i. e.*, at the 6400-ft. level in the hypothetical mine shown in Fig. 7. This course would entail some sacrifice of the "positional efficiency," but I believe that in most cases it would be compensated for by savings in power and cost and by greater simplicity.

Mr. Waterfill refers to the condenser water cooler at the upper level as a humidifier, which is, of course, a strictly accurate description: however, in Fig. 9 this apparatus is marked "dehumidifier," and the water pipes leading to and from it are shown as "pipes to dehumidifier." This is a small matter but I think it is worth pointing out, as it might possibly tend to confuse anyone not fully conversant with these problems.

The system which Mr. Waterfill describes has already been successfully employed for some time in conditioning the large air volumes used in the ventilation of cinema theaters and other public buildings, and there would seem to be no reason why it should not meet with similar success when applied to mining.

R. W. Waterfill (written discussion).—Mr. Davies' suggestions are of real value in increasing the practical value of the paper. In order to make the paper concise many important details were omitted. As a great deal of explanation might be necessary to make sure that no misunderstanding of the conditions on which the horsepowers, etc., are based, such figures were avoided. The approximate horsepowers required to operate the hypothetical plant illustrated in Figs. 7, 8 and 9 are given below. It is to be remembered, however, that even in this arrangement, variations in water velocities, number and type of pipe fittings, efficiencies, etc. materially affect the results. The brake horsepowers are estimated as approximately:

1. Refrigerating machines	300
2. Cold-water pumps	50
3. Condenser-water pumps	130
4. Dehumidifier fan	15

Item 3 could be considerably reduced by placing the condenser water cooler approximately on the same level as the other equipment, wherever this is possible. The actual pumping power may be made as low as 25 hp. Item 3 illustrates the point previously made regarding the possibility of misleading conceptions resulting from attempts to give such data in an article of this kind.

The term humidifier should be used in all references to the condenser water cooler unless perhaps we should speak of it simply as the condenser water cooler. Fig. 3 illustrates the dehumidifier used in cooling the air supplied to the working places. The humidifier is of the same general appearance. The terms humidifier and dehumidifier are chiefly descriptive of the application rather than appreciable differences in equipment. The former indicates that the purpose is to add water vapor to the air passing through it; the latter, to remove water vapor from the air.

Ventilation at United Verde Mine

By Oscar A. Glaeser,* Jerome, Ariz.

(New York Meeting, February, 1929)

The mine and plant of the United Verde Copper Co., located in Yavapi County, at Jerome, Arizona, have been described in various technical publications and, therefore, a brief outline of its essential features will suffice. The orebodies are of the schist replacement type, the main sulfide mass being a large lens-shaped body approximately 7 acres in cross-sectional area. In general, the mineralization is found along the iron-schist contact, with the orebodies extending about 1000 ft. along this contact and varying from a few feet to 250 ft. in width. The mine is located on a mountain slope and has openings into it on three horizons, as follows:

1. Raises and ore passes open vertically into the shovel pit. These extend down to the 1000-ft. level, where they terminate as loading bins in the Hopewell adit. The main fan intake air shaft, which also extends vertically down to the 1000-ft. level, has its surface opening on the hillside at the extreme north end of the shovel pit, and the main mine air return exhausts through a raise at the extreme south end of the shovel pit.
2. The 500-ft. level adit.
3. The Hopewell adit on the 1000-ft. level. The mine has a vertical downward extent of 2000 ft. below the 1000-ft. level.

Underground operations are carried on through two main shafts, both of which are wholly underground. Shaft 5 is the ore-hoisting shaft. It has three compartments and extends from the 800 to the 3150-ft. level. Shaft 6 is the service shaft, and is used for lowering and hoisting men and material. It has a large cage compartment and three smaller compartments for counterweights, ladderway, and pipe and power lines. This shaft extends from the 500 to the 2550-ft. level, and is now being extended to the 3000-ft. level.

An air shaft for the downward movement of fresh air from the fan has been provided, and extends to the 2550-ft. level at this time. Work has been started to extend this air shaft to the 3000-ft. level. A return airway for vitiated mine air, in the form of raises and short connecting drifts, extends downward to the 2100-ft. level.

The underground mining operations extend from the 300 to the 3000-ft. level. There are about 45 stopes in active operation on the 17 levels above the 2400-ft. level and approximately 24 development faces are being carried forward at this time. There are also about 16 miles of open drifts and crosscuts, all of which must be served in the general scheme of ventilation.

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* Ventilation Engineer, United Verde Copper Co.

The ore-extraction methods, in the order of their importance, are: horizontal cut and fill; square set; top slice; inclined cut and fill; timbered rill; shrinkage.

NECESSITY FOR MECHANICAL VENTILATION

Mechanical ventilation is utilized at the United Verde mine to overcome conditions incidental (1) to mining operations and (2) to the depth at which operations are now carried and (3) to keep cool all workings adjacent to a sealed fire area.

The main objects to be accomplished are, first, to maintain a healthy atmosphere with a maximum oxygen content; second, to keep down the temperature and relative humidity of the mine air; and, third, to maintain an air current which will sweep all workings free of blasting fumes.

The atmospheric standards aimed at are: temperatures in working places not to exceed 80° F.; relative humidity in working places not over 80 per cent.; air velocity 125 to 150 ft. per min.; volume of air per man (minimum) 250 cu. ft. per min. 80° F. may appear to be a high temperature; however, if sufficient air is circulated so that the temperatures in the bottom development faces, where the rock temperature is 95° F., even approaches 80° it is found relatively easy to keep below that temperature in other workings where definite air currents have been long established.

Air velocities are as a rule difficult to maintain where most needed such as at development faces and in stopes. Velocities of 1000 ft. per min. may be maintained in manways and raises leading into stopes, and yet the velocity in the stopes may be relatively slight, depending, of course, on the open area of the stope. Velocities of over 500 ft. per min. are too high for ladder manways, since it is difficult to keep a light, and dust in eyes is a common occurrence which may at any time be translated into an accident. Where large volumes of air must be passed through a drift or raise, high velocity becomes a necessary evil. Such cases are the exception rather than the rule.

The desired absolute minimum for each worker is 250 cu. ft. of air per min. Practically all working places have an air circulation far in excess of the minimum. In fact, the actual volume of air delivered into the mine is equivalent to about 740 cu. ft. of air per man per minute.

Effect of Surface Temperature on Mine Air

All of the air handled by the main fan on the 1000-ft. level (Fig. 1) is drawn from the surface through a shaft 980 ft. deep and 169 sq. ft. in

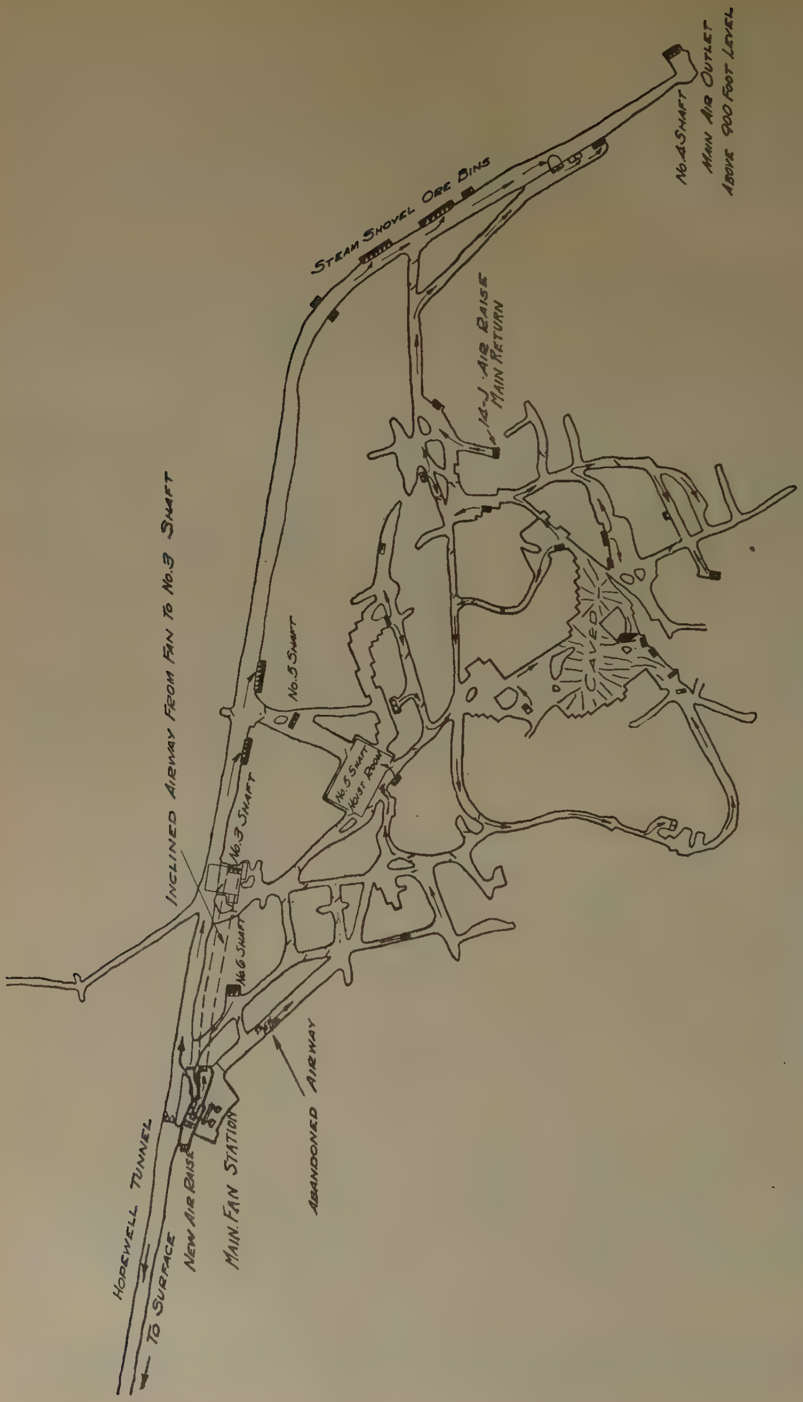

FIG. 1.—PLAN OF 1000-FT. LEVEL SHOWING VENTILATING SYSTEM.

section area. Under present fan-operating conditions the air will traverse the entire length of this shaft in 40 sec. The temperature of surrounding rock can have but little influence on the temperature of the air moving in the shaft. In Table 1, under the August temperature column, it will be noted that there is a drop of 7° F. from the surface to the fan intake, and a rise of 4° F. in the fan; the latter is due to heat given off by the electric appliances and to friction within the fan. The temperatures taken in October at the same points would indicate that the rock temperature of the intake shaft is in the neighborhood of 70° F., as there is practically no difference in the air temperature between the surface and the fan intake. Under existing summer conditions, the mine workings absorb heat from the air when in reality the reverse should be the case. Cooling development faces in ground temperatures of 95° F. with air, which to begin with is 82° F., has been found difficult. If, however, a sufficient volume of this air with a low moisture content can be brought to a working face with a velocity of 150 ft. per min. or higher, the cooling effect on the men will be considerable. Just how the high temperature of the intake air can be reduced without the installation of a costly refrigeration plant is not known, since the summer of 1928 was the first hot season under the new arrangement. Experiments with cold-water sprays have been tried on a small scale, and these will be continued in the future.

TABLE 1.—*Temperature and Humidity of Mine Air*

Mine Fresh Air Supply		Aug. 20, 1928		Oct. 22, 1928	
Level, Ft.	Volume, Cu. Ft. per Min.	Temperature, Deg. F.	Relative Humidity, Per Cent.	Temperature, Deg. F.	Relative Humidity, Per Cent.
Surface atmosphere.		92	13−	70	22
Intake side of fan.		85	16	70	22
Discharge side of fan.		89	13	71	20
		—	—	—	—
Difference due to fan.		+4	−3	+1	−2
1,000	20,000	89	13	71	20
1,200	37,200	89	13	71	20
1,350	9,000	89	13	71	20
1,500	8,000	89	13	72	21
1,650	38,600	89	13	72	21
1,800	45,000	88	15−	73	22
1,950	8,200	87	19	73	22
2,100	3,000	87.5	19.5	73	25
2,400	48,000	82	27.6	73	28

Heat and Humidity Due to Rock Temperature and Mine Water

Rock temperatures due to depth are at this time a factor only below the 2400-ft. level. Temperatures taken in drill holes and of mine water on the 3000-ft. level were uniformly 95° F. Very little ground water has been encountered so that most of the moisture is the result of wet drilling, and, because evaporation is quite rapid at these temperatures, the humidity is rather high, averaging 85 per cent. As accurately as can be determined the rock temperature on the 2400-ft. level is 78° F., and on the 300-ft. level 65° F.

Effect of Sulfides on Air

It is probable that stopes in solid sulfides require more air than do stopes in any other kind of rock. The air leaving such stopes is generally warmer than that leaving other stopes. As a rule the humidity is also higher and, further than that, the air has a faint but characteristic musty smell. All of these conditions can be eliminated almost entirely by increasing the air circulation through such stopes.

Heat Radiation from Sealed Fire Area

A portion of the sulfide mass is termed as the "fire area" inasmuch as certain areas in it are burning and have been for some 35 years.

The fire area is almost wholly in a block of ground 500 ft. long, 300 ft. wide, and reaching from the surface to the 800-ft. level. This area has a tremendous heating capacity. Temperatures taken behind various bulkheads range from 170° to 190° F. Temperatures in open drifts passing this area were formerly 90° to 95° F.; through better ventilation these temperatures have been reduced to 75° F. The areas in which ore is now being mined on the upper levels are some distance from the actual fire zone and are not affected by it.

Mine Gases

The massive sulfides of the United Verde mine contain as high as 46 per cent. sulfur. Blasting in this "hard iron" frequently causes the formation of sulfur dioxide (SO_2) and hydrogen sulfide (H_2S), both of which are exceedingly dangerous. A concentration of 0.05 per cent. SO_2 is dangerous upon exposures of longer than 30 min.; while a concentration of 0.06 per cent. H_2S will cause unconsciousness within 2 min. and death within 15 min. Samples of these gases were taken in a heavy sulfide drift 15 min. after blasting, and analyzed by the U. S. Bureau of Mines.[1] The analyses indicated 0.09 per cent. SO_2 and 0.07 per cent. H_2S.

A system of electric signals has been installed on the various levels below the 2100-ft. level. The order of blasting is downward from level to level. When the men on one level have spit their shots and passed the signal station the signal is sent down to the next level. Word is then sent out on that level to light their shots and so on down. This system, when strictly followed, eliminates all chances for men being caught in an ascending current of gassy air. The men below the level which is blasting at any particular time are protected against gases coming down on them only by ascending air currents. The necessity of keeping all air currents upcast is of great importance.

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[1] E. D. Gardner, G. W. Jones and J. D. Sullivan: Gases from Blasting in Heavy Sulphides. U. S. Bur. Mines *Rept. of Investigations* 2739 (March, 1926).

Blasting fumes or powder smoke are of little or no importance because practically all heavy blasting is done at the end of the shift.

Dust

In general, dust has not been a problem throughout the mine. This is due chiefly to the fact that dry drilling is prohibited. Dust has required attention, however, at the loading bins in the Hopewell adit and at transfer points in waste raises and ore grizzly chambers. Strong air currents are used at these points to keep the dust back and water sprays have proved effective in settling dust.

Decay of Mine Timber

Considerable attention has been directed to the possible prevention of decay of mine timber. Where fungus growths once flourished, good ventilation has brought about a healthy timber condition which is unquestionably an aid in reducing the cost of timber and timbering in the mine.

Main Air Courses

The main air courses may be classified into three groups: (1) the fan intake shaft (known as 19-R air raise); (2) the main air delivery shaft; (3) the main return airway. The fan intake shaft is the surface connection to the intake side of the fan, and was driven solely for this purpose. The shaft has an effective cross-section area of 169 sq. ft., and is approximately 980 ft. deep. The walls are supported by concrete rings 3 feet wide, spaced 4 ft. apart. The saving over a solid concrete lining is estimated at 40 per cent. Taking all factors into consideration it was felt that the air requirements for years to come would be such that there would be relatively little loss due to friction. The maximum velocity at this time is 1500 ft. per min. There are no connections to the intervening levels, except one; a small drift was driven on the 500-ft. level, from which the work of raising to the surface was conducted. Because the shaft is under a negative air pressure, and in order that the fan would at no time draw air from the 500-ft. level, two iron doors, set in concrete frames, have been placed in this drift. The outer door is kept locked at all times.

Shaft three, which was originally a hoisting shaft extending to the 1950-ft. level, has been converted to the main air delivery shaft. A concrete bulkhead 5 ft. thick was put in the shaft about 20 ft. above the 1200-ft. level. An inclined raise, 300 ft. long, connects this shaft immediately below the bulkhead with the outlet side of the main fan. The shaft was stripped of all timber. Concrete walls were built at all level intersections with a 4 by 6-ft. iron regulator door at each station, which can be quickly adjusted for any desired quantity of air. The shaft is in solid diorite requiring no support, and therefore it was left unlined with an effective open cross-section area of 80 sq. ft. Slightly inclined raises of the same cross-section area connect all levels from the 1950 to the 2400 ft. levels, inclusive. Fresh air is taken off on all levels, beginning with the 1000-ft. level and ending on the 2400-ft. level. It will be noted from Table 1 that the volumes taken off vary greatly on different levels. The reason for this will be discussed later; attention is called to it here merely to point out the importance of dependable regulators on all the levels by means of which the amounts taken can be adjusted to suit the needs.

The main return airway consists of a series of raises and connecting drifts of various lengths, beginning on the 2100-ft. level and extending upward to the 900-ft. level. At this point it connects with the old shaft 4 which extends to the 300-ft. level. A drift and a short raise to the surface complete this return air course. All raises and drifts in this course are stripped and free of timber. The shaft was stripped and lined with concrete rings. All mine openings driven to complete this return air course have an effective open cross-section area of 120 square feet.

At the present time air enters the return on the 2100, 1500, 1350, 1200 and 1000-ft. levels. Regulators are not necessary at the junction points. The full size is maintained at the intersection so as to offer as little resistance as possible to the flow of the air.

AIR DISTRIBUTION AND REGULATION

As in most metal mines, the United Verde has its main haulageways with drifts and crosscuts branching off in various directions and at varying intervals. Chute raises and manways into stopes are put up wherever necessity demands.

The air shaft discharges the fresh air through a regulator into the main haulageway. The intake air then splits at the drift intersections, spreading over the entire level. Wherever an opportunity is afforded, it moves upward. Thus the stopes nearest the source of supply have a tendency, if not controlled, to take most of the air, leaving but little for the stopes located at the far ends of the orebody. Much work has been done to overcome this uneven distribution, and the results have been reasonably and uniformly satisfactory.

All stopes have raises through to the level above and inasmuch as all the air entering the stopes must pass through these raises, the best control is obtained at the top of the raises. Frames are set into these raises, and solid doors, or open mesh covers hinged to the frame, or a combination of the two is used, depending on the amount of air desired. This method has been found to be far more effective than regulation from below. Ordinarily the drift cannot be shut off by a door or regulator; as a rule, other stopes are on the same drift and these too must be served. Putting trapdoors in manways is not effective because chutes are constantly being drawn and any number of them may be empty at the same time, thus causing a large loss of air.

Doors have been freely used to control air currents on the levels and, while many of them have been installed knowing that no immediate use would be made of them, nevertheless, in practically every case their installation has proved necessary and valuable in that they were ready for use when most needed. One of the important features of the ventilation system at the United Verde mine is that all air currents are, or should be, upcast after leaving the level drift which conducts them from the air shaft. It is one of the first duties of the ventilation department to keep them upcast. It frequently happens that when a raise is holed through, or a drift breaks into another, the air currents change in some district and become downcast. Such a condition must ordinarily be righted before the next blasting period in order that no gases move down on the men below. It is at such times that doors previously installed but not used serve their purpose. The fundamental cause for such reversal is then studied and adjustments made accordingly.

One frequently hears of mines having good air in the drifts, but hot and humid conditions prevailing in the stopes. This condition is not necessarily the fault of the general ventilation system but is often due entirely to the mining methods which provide inadequate entrances to the stopes from below or outlets to the level above, or a combination of the two. It is at best a difficult problem to establish proper circulation over the entire stope. A miner may be working in some corner and be uncomfortably warm, and yet there may be from 5000 to 10,000 cu. ft. of air per min. circulating in the stope. In this case the probable trouble is that the air comes up the manway and takes the most direct route to the raise going to the level above.

In coal mines the mining methods are such that, with the aid of brattice lines, the air is taken to the working face. In this case it is of vital necessity, not so much to keep the miner comfortable as to sweep away dangerous gases. In metal mines these explosive gases generally do not exist, hence, no brattice lines are used. Besides, with the back from 8 to 20 ft. above the muck pile, which in itself is very irregular, and with the heavy blasting necessary in this as well as most metal mines, it is almost impossible to carry brattices in stopes. At the United Verde mine considerable attention is paid to stope ventilation during the development stages. Manways are so located that the far extremities get fresh air from below. Ventilation requirements are also considered when locating waste raises to the level above, so that the stopes have a reasonable distribution of air through them.

Ventilation of Dead Ends

Dead-end drifts and crosscuts are ventilated by auxiliary fans and tubing. Exploration work below the 2400-ft. level is carried on from a winze in the ore zone. A Sturtevant No. 10 blower, located at the collar of the winze on the 2400-ft. level, supplies fresh air to six working faces on three different levels. The air is taken down through 16-in. galvanized iron pipe, with a 16-in. branch on each level. A "T" at the end of the branch splits the air sending it in opposite directions. The pipe is reduced at this point to 12-in. Flexible tubing is used at the end of the metal pipe; 50 to 100 ft. of tubing, known as the "blasting end," is in use is each drift. When the volume of air delivered at the end of the tubing is too small to properly ventilate the face, a compressed air-driven booster is installed in the line. Should the work in any drift stop, the tubing is removed and a metal cap placed over the end of the pipe, thus saving that air for some other drift. When the fan at the top of the winze on the 2400-ft. level was installed, the piping was so arranged that with four gates the air current could be reversed and the air drawn out of the winze levels. It was felt that this might be necessary in order to clear the faces from blasting fumes in time for the next shift to go to work. This practice has not been necessary, however, and the fan is used for blowing only.

Another Sturtevant fan of the same size as the one mentioned above is being used as an exhauster in a 2100-ft. drift which, at this writing, is about 1750 ft. in length. The fresh air moves toward the face of the drift. It is picked up by an air-driven blower about 20 ft. outside the suction end of the exhauster pipe and sent to the face through flexible tubing 100 to 250 ft. long. A curtain between the blower and the suction end of the exhauster line has not been found necessary. Other drifts are ventilated by small electric units or compressed air-driven blowers. In all cases metal pipe or flexible tubing is used to direct the air.

Air Splits

Splitting of the intake air, or putting various parts of the mine on independent air splits, is demanded by law in the coal mines of most of the

TABLE 2.—*Stope Temperatures and Relative Humidities*

Level, Ft.	Average Temperature, Deg. F.	Average Relative Humidity, Per Cent.	Condition of Air*
September to November, 1925			
1800	76.0	93.1	
1950	77.5	94.0	
2100	76.5	88.5	
2250	77.0	84.6	
2400	78.0	87.5	
Oct. 15 to 18, 1928			
300 to 600, incl.	64	79	*A*
700 to 900, incl.	76.5	84.3	*B*
1200	66.5	62.0	*A*
1500	71.5	90.4	*C*
1650	68.5	80.4	*B*
1800	68.5	79.4	*A*
1950	74.0	88.4	*C*
2100	79.0	88.0	*C*
2250	73.5	79.0	*B*
2400	74.0	58.3	*A*

* *A* indicates fresh air which has not been used; *B*, air previously used in one or two working places and freshened by addition of fresh air; *C*, vitiated air, used because there is no fresh air available, or because no other air course is available by which it might be removed.

TABLE 3.—*Air Accumulation into Return Airway*

Level, Ft.	Volume, Cu. Ft. per Min.	Temperature, Deg. F.	Relative Humidity, Per Cent.
2,100	40,000	76.0	93.1
1,500	21,300	75.0	95.5
1,350	10,400	76.5	90.5
1,200	15,900	78.0	100.0
1,000	12,000	76.0	100.0
Total	99,600		
Through orebins, above 1,000 ft.-level	85,000		
Through shaft 5, bottom level ventilation	35,000		
Loss through shafts, open doors and leakage	35,400		
Total	255,000		

states. That such a system exists in a metal mine is the exception rather than the rule.

Separate air splits for various parts of the mine were established during 1928. As shown in Table 2, under the column "Grade of Air," letterr from *A* to *D* and combinations thereof indicate the condition of the ais

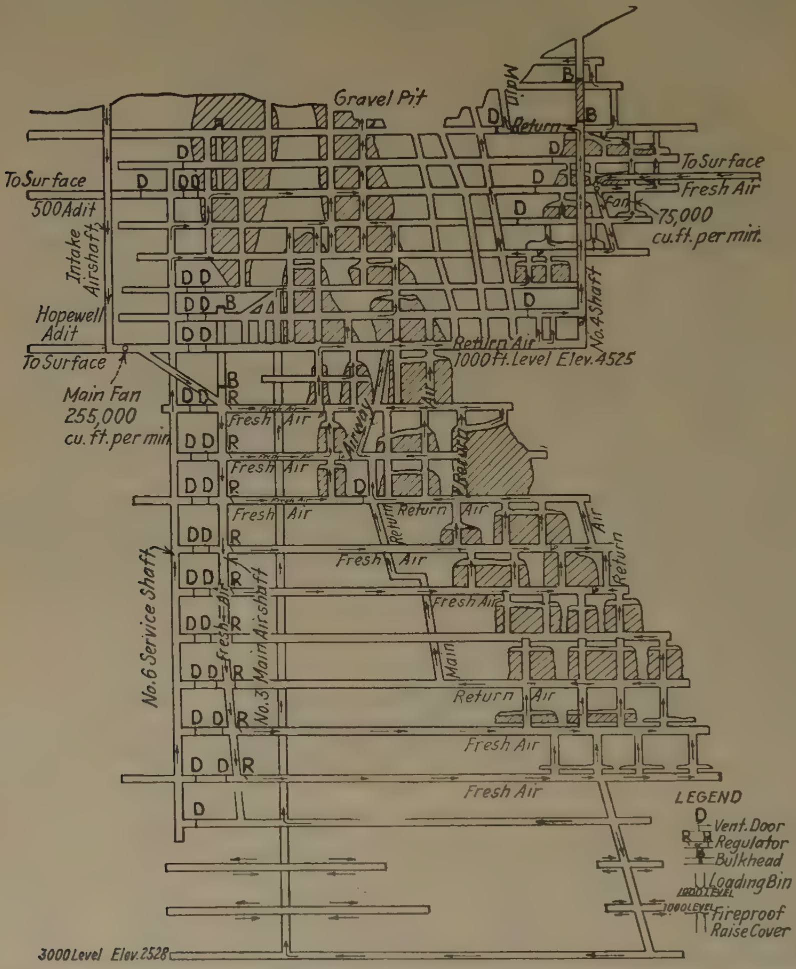

FIG. 2.—SECTION THROUGH UNITED VERDE MINE SHOWING GENERAL SYSTEM OF VENTILATION.

entering that particular stope. All grade *A* air comes direct from the fan; grade *B* has been used but once or passed through but one working place prior to its reaching the stope in which it is noted; grade *C* has been used in two places; and grade *D* having been used in three places, is considered vitiated and is directed to the return by the shortest possible route. Only a few stopes are ventilated by the so-called vitiated air.

The fact that they are far removed from fresh air makes this necessary. A number of other stopes have used air of various conditions flowing through them. In most of these stopes this more or less vitiated air is freshened by the addition of intake air so that results are satisfactory.

For a clearer picture of air-splitting as practiced at the United Verde mine, reference is made to Table 1 and Fig. 2. From the table it will be seen that all levels get some fresh air, and that the amounts delivered to the 1350, 1500, 1950 and 2100-ft. levels are less than the volume delivered to the other levels. These are the gathering levels of the return air system.

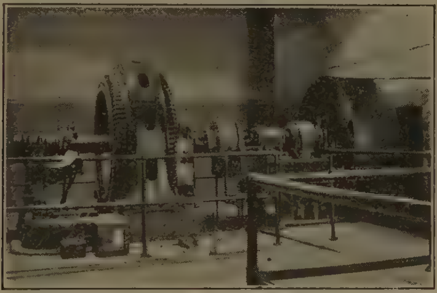

FIG. 3.—MAIN FAN DRIVE, SHOWING 400-HP. DIRECT-CONNECTED MOTOR, DRIVE SHAFT CLUTCH, AND BELT CONNECTION.

Instead of releasing the fresh air on the various levels and allowing it to rise through the workings to the surface, its course is carefully controlled and, after the air has ventilated a number of working places and has become unfit for further use, it is withdrawn to the return air raise (Table 3).

MAIN FAN

The main fan is a Jeffrey, double inlet, reversible type, with forward curved blades. (Fig. 3.) It was put into operation in October, 1927. The rotor is 4 ft. 6 in. wide by 9 ft. dia. It is direct-connected to a G.E. 400 hp. 2200-volt, 280-r.p.m., 3-phase, 60-cycle, slip ring motor. A Westinghouse, type CW, wound rotor induction motor of 350-hp., 2200-volt, 3-phase, 60-cycle, and 350 r.p.m. is used as an auxiliary drive. It is belt-connected and can be called into service on a moment's notice by means of a clutch on the fan drive shaft. Both motor controls are on one switchboard. (Fig. 4.) A pushbutton switch cuts the power from one motor to the other. The speed of both motors can be changed at will by means of a pushbutton type control, with a bank of resistances in the secondary circuit arranged to give three speeds.

Variable fan speeds have been found invaluable at this mine particularly during blasting periods when it is desirable to slow up the air currents. To stop the fan entirely is considered dangerous because of the reversal of air currents in certain parts of the mine. The fan is therefore operated at low speed during these periods. A considerable saving is also effected by operating the fan at low speed on the graveyard shift, on Sundays and other holidays when the mine is not on a production basis. At such times the slow speed belt drive is used. At the present time the fan is operated at second speed on the direct drive on the day and afternoon shifts only, since full speed operation is not required under present conditions.

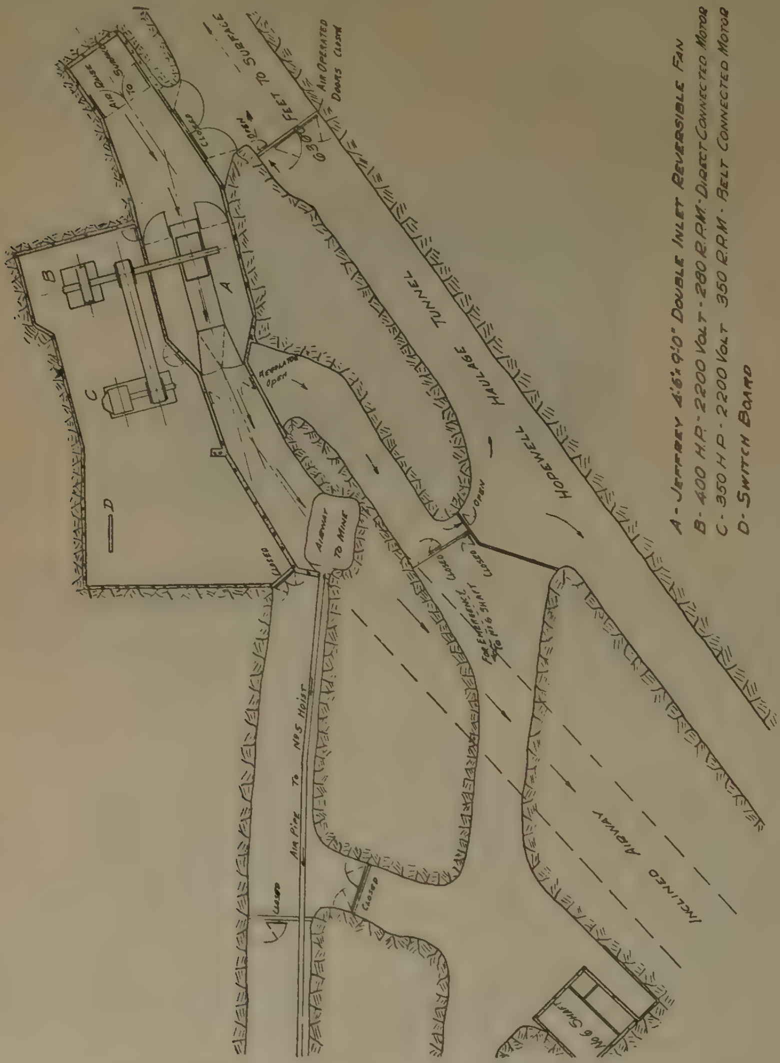
A - Jeffrey 4'6" x 9'0" Double Inlet Reversible Fan
B - 400 H.P. - 2200 Volt - 280 R.P.M. - Direct Connected Motor
C - 350 H.P. - 2200 Volt 350 R.P.M. - Belt Connected Motor
D - Switch Board
Air Raise to Surface
Closed
Open
To Surface
Feet
Air Operated Doors Closed
Tunnel
Haulage
Hopewell
Regulator Open
Airway to Mine
Closed
Open
For Emergency to No 6 Shaft
Air Pipe to No 5 Hoist
Inclined Airway
No 6 Shaft
A
B
C
D

FIG. 4.—NEW FAN STATION ON 1000-FT. LEVEL.

FAN PERFORMANCE

When operations were extended some years ago to the present lower levels, it became evident that the old ventilating system and its airways would have to be improved. A series of tests were made on the fan installation in 1925. The results of these tests are briefly tabulated in Part *A*, Table 4. The fan, a ⅔ width, single inlet, Sirocco, was carrying a load for which it was not designed. Considerable mechanical trouble was experienced from time to time because of the light construction and also because it was being operated at a much higher speed than that for which it was designed. A new fan installation was, therefore, decided upon.

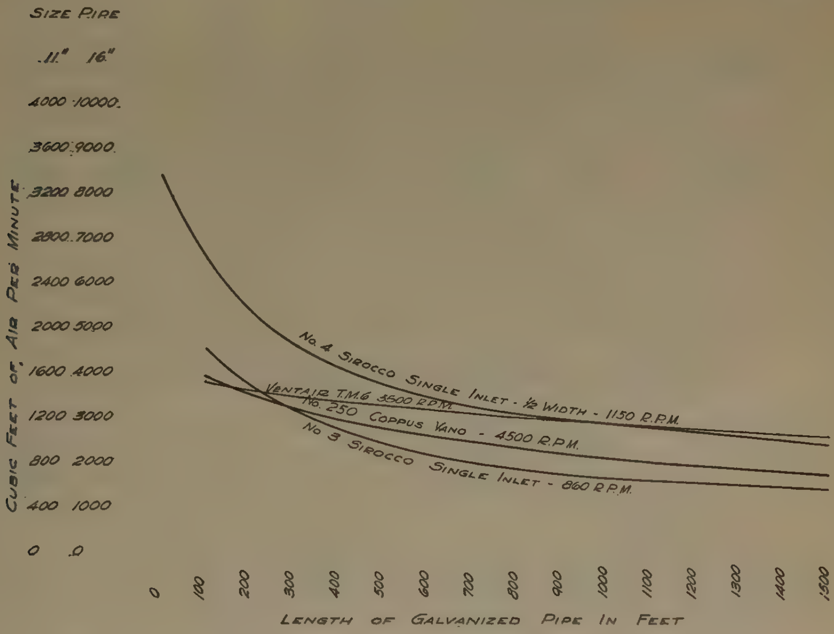

FIG. 5.—CAPACITIES OF AUXILIARY FANS USED AT UNITED VERDE MINE.

Future requirements were placed at 250,000 cu. ft. of air per min., at 5.3 in. pressure. After this fan was installed and placed in operation some tests were made on it. It was, of course, known that the fan would not operate under present conditions at its highest efficiency because the mine resistance has not been changed materially. It is expected to increase, however, when the air shaft is extended to the 3000-ft. level. Tabulation *B* gives briefly the performance figures under present conditions. While the overall efficiency is not high, a much greater volume per horsepower is now being delivered than formerly.

The improved method of handling the return air has unquestionably been of considerable help in this connection. Tabulation *C*, Table 4, gives performance data under conditions for which the fan was designed. This condition was obtained by closing the fan discharge until a pressure of about 5.3 in. was obtained.

AUXILIARY FAN EQUIPMENT

The two No. 1 Sturtevant blowers have already received mention. Their field of application is limited and they are, therefore, not considered as standard auxiliary equipment. (Table 5.)

TABLE 4.—*Fan Performance Data*

A, Old Main Fan, ⅔-width Sirocco, Rotor Dia. 7 Ft. 6 in. by 2 Ft. 6 in., Belt-drive, 400-hp., 280-r.p.m. Motor, October, 1925

Speed	Fan, R.p.m.	Volume, Cu. Ft. per Min.	Pressure, In.	Horsepower		Volume per Hp., Cu. Ft. per Min.	Overall Efficiency, Per Cent.
				Theoretical	Actual		
Full	352	191,000	3.56	107	428	446	25.0
Second	287	164,000	2.62	68	393	417	17.3
Low	220	132,000	1.70	35	359	363	10.0
B, New Fan under Actual Operating Conditions, June 17, 1928							
Full	268	297,500	3.6	169	438	679	38.5
Second	233	255,300	2.6	105	326	783	32.0
Low	176	182,500	1.6	46	195	936	23.4
C, New Fan Operating under Specification Conditions							
Full	274	255,000	5.33	214	433	589	56
Second	237	222,000	3.74	131	342	649	38

The No. 3 Siroccos are old fans; they are direct connected to a 7½-hp. motor, and in average mine service they will develop a maximum pressure of 3 in. While they are still being used, their service is confined to work requiring less than 400 ft. of 16-in. pipe.

A No. 4 Sirocco is giving good service on a 16 in. line 700 ft. long. Under this condition it delivers 3500 cu. ft. of air at 4.5 in. pressure.

The disadvantages of the units mentioned lie in their size and weight, as well as the low pressure they develop. The No. 3 can ordinarily be transported as a unit, although four men are required to handle it. The No. 4 weighs 1150 lb., and must be sectionalized for transportation.

When it became necessary to purchase more fans, it was decided to find a fan which did not have the disadvantages mentioned above. Thus,

TABLE 5.—*Auxiliary Fan Equipment*

Make	No.	Type	Size	Motor, Horsepower	R.p.m.	Volume, Cu. Ft. per Min.	Pressure, In.	Diameter of Duct	Length, Ft.
Sturtevant	1	Straight blade	10	25	865	8,600	4.6	16-in. pipe	450
Sturtevant (suction)	1	Straight blade	10	40	850	3,200	8.0	16-in. pipe	1500
American Blower	1	Sirocco	4	10	1150	See Fig. 5			
American Blower	4	Sirocco	3	7½	860	See Fig. 5			
Coppus	4	Ventair	TM-6	5	3500	See Fig. 5			
Air Blowers, Compressed:									
Coppus	1	Vano	SP350		4000				
Coppus	12	Vano	SP250		4500				
	6	Injector*	12 in.						

* Made in United Verde shops.

Comparison of Rotating and Injector Types of Blowers*

Type	Nozzle Dia., In.	Compressed Air, Cu. Ft. Free Air per Min.	Volume Delivered 200 Ft.-Ventube	Cost
Rotating	¼	50	2700	$261.00
Injector	3/16	44	2100	8.30
	¼	54	2700	
	⅜	100	3200	

* Tests made with compressed air at 90 pounds.

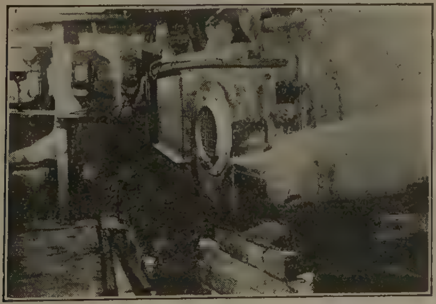

FIG. 6.—AN AUXILIARY FAN INSTALLATION (VENTAIR TM-6) ON THE 2850-FT. LEVEL OF UNITED VERDE MINE.

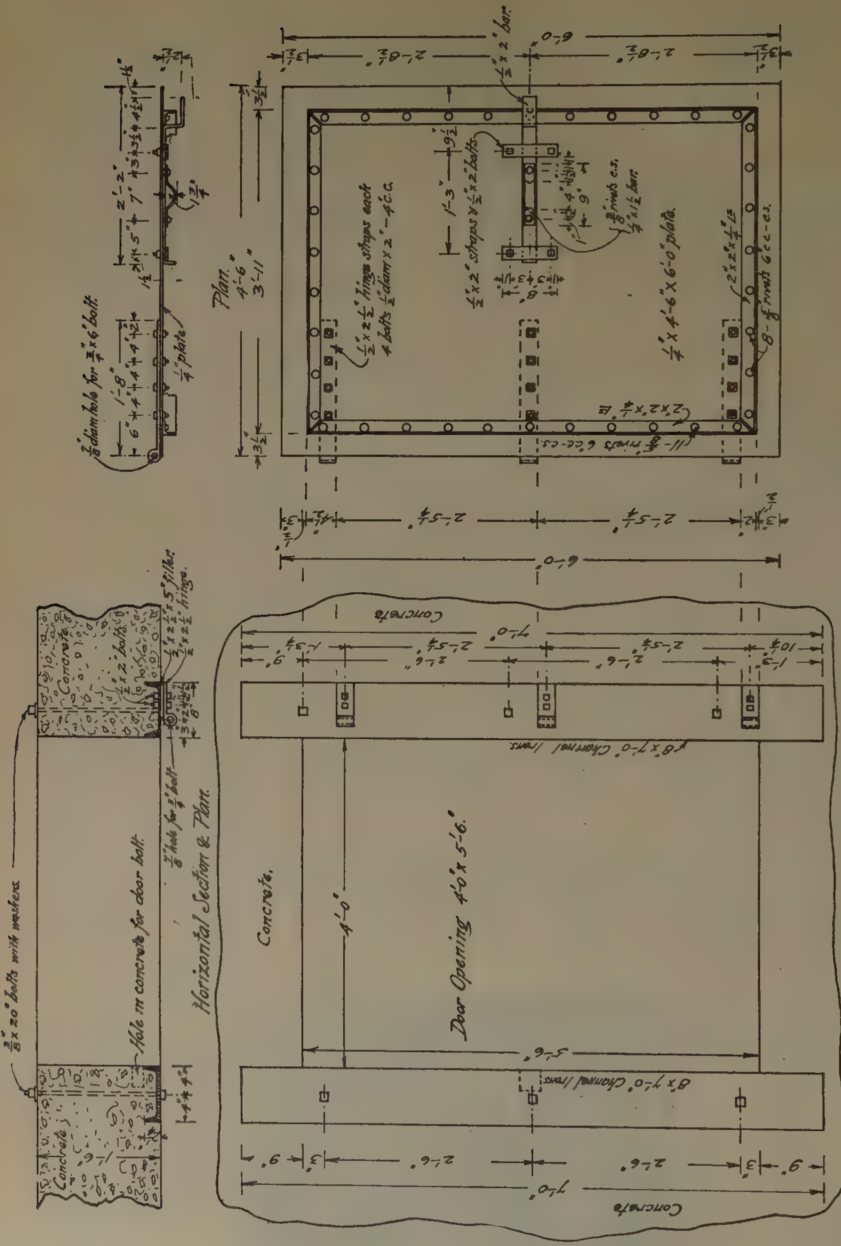
Plan.
4'-6"
3'-11"
6'-0"
2'-8 3/4"
2'-5 3/4"
1/2" x 2" bar.
1/2" x 2" straps & 1/2" x 2" bolts
1'-3"
9 1/2"
3/8" rivets c.s.
1/4" x 1 1/2" bar.
1/2" x 2 1/2" hinge straps each 4 bolts 1/2" diam x 2" - 4" c.c.
1/4" x 4'-6" x 6'-0" plate.
2" x 2" x 1/4" L's
8 - 3/8" rivets 6" c-c- c.s.
2" x 2" x 1/4" L's
11 - 3/8" rivets 6" c-c- c.s.
7/8" diam hole for 3/4" x 6" bolt
1'-8"
2'-2"
1/4" plate
Horizontal Section & Plan.
3/8" x 20" bolts with washers
Hole in concrete for door bolt
7/8" hole for 3/4" bolt
1/2" x 2" bolts
1/2" x 2 1/2" x 5" filler
1/2" x 2 1/2" hinge
Concrete
Concrete.
7'-0"
2'-6"
8" x 7'-0" Channel Irons
4'-0"
5'-6"
Door Opening 4'-0" x 5'-6"
Elevation Door Opening

FIG. 7.—CONCRETE AND STEEL FIRE DOOR.
Easily removable track should be taken out before fire door can be closed.

the Ventair TM-6 was chosen and three of these units have now been in constant use for 4 months (Fig. 6). Thus far the results have been satisfactory. These fans weigh about 450 lb. They are small enough so that they can be handled on any mine truck or small cage without being dismantled. Since they are of the backward curve blade type, their characteristics are such that the volume of air delivered is maintained through much greater lengths of pipe or tubing than is possible with the other types of small fans. They are direct-connected to a Louis Allis, 5-hp., 230-volt, D.C., 3500 r.p.m., semienclosed, shunt-wound motor, and are rated to deliver 3000 cu. ft. of air per min. against a resistance of 6 inches.

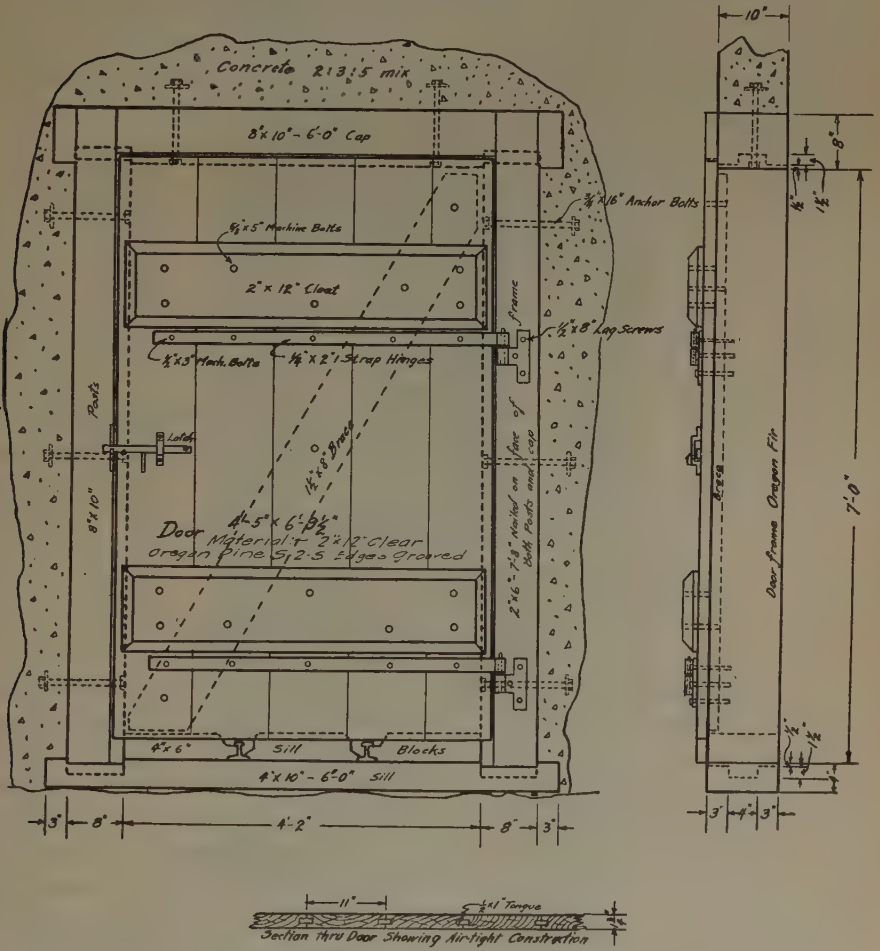

FIG. 8.—UNITED VERDE VENTILATION DOOR (IMPROVED TYPE).

A Coppus Vano blower, SP-350, is part of our fire-fighting equipment, and is held in reserve for that purpose.

Twelve Vano blowers, SP-250, are used where a set-up for only a short period of time is desired. They are easy to install, and require only a common air connection. While they are primarily a low-pressure blower, they nevertheless deliver 2700 cu. ft. of air through 200 ft. of 12 in. flexible tubing. These blowers are now, however, being gradually replaced by an injector type blower, which has been successfully used in the United Verde mine for a period of a year. These blowers contain no rotating parts and have consequently no maintenance expense connected with their operation. It is an ideal line booster and is successfully used in ventilating drifts and raises up to 300 ft. in length.

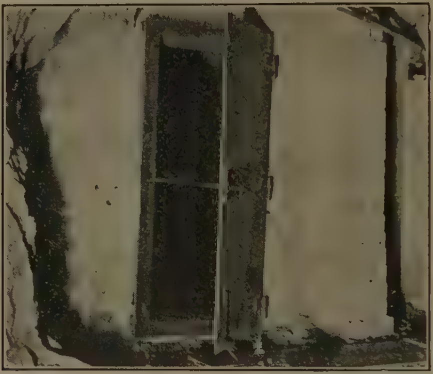

FIG. 9.—AN IRON DOOR ADJUSTABLE SCREW TYPE REGULATOR FOR VENTILATION CURRENTS.

VENTILATION ACCESSORIES

The mine doors are of two types, namely, the ventilation door, which is of wood construction, and the iron fire door, which is used for fire control purposes. The latter is installed only where a door of non-combustible material is considered necessary. Twenty of these doors are in use. Most of them are located in the approaches of worked-out timbered stopes, so that these areas can be effectively shut off from the rest of the mine. These doors are always hung to open inward so that in case a concrete seal becomes necessary, the door can be used as the inside form for the concrete. All iron doors are hung in a concrete frame. The latches are connected to levers on both sides of the doors so that they can be opened from either side.

The control of the air in its flow through the mine depends largely on the use of doors. These prevent the air from following the path of least resistance. An absolutely tight door is impracticable but, on the other hand, leakage through doors can be held to a minimum by, first, the proper design and installation, and second, by proper upkeep after installation. The doors shown in Figs. 7 and 8 are used in the United Verde mine. They are hung from a wooden frame set in concrete and are all equipped with latches and counterweights to keep them closed. Where only small volumes of air are desired, necessitating constricted drift openings, doors containing slide regulators are used (Fig. 9).

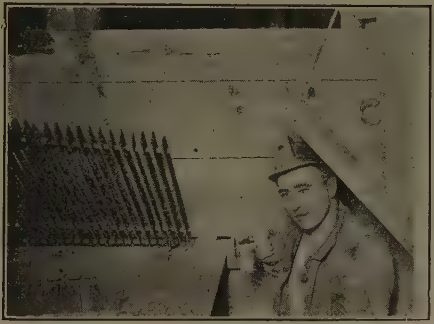

FIG. 10.—GRID COVERS OVER MANWAY AND TIMBER SLIDE.

Manways into stopes were formerly covered with a hinged wooden cover as a safety measure to prevent the falling of material. This practice has been changed, however, and now a standard size hinged iron grid cover is used. It offers but little resistance to the flow of air and gives good protection. Covers over raises coming through from a stope below have already been mentioned and are shown in Figs. 10 and 11.

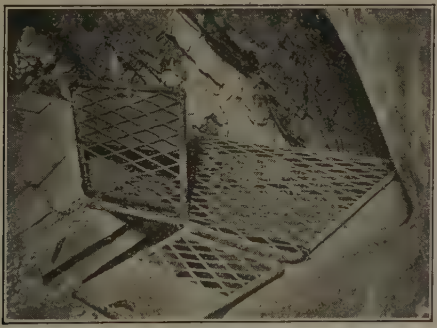

FIG. 11.—RAISE COVER OF EXPANDED METAL; THIS AND GRID TYPE OF COVER (FIG. 10) OFFER BUT LITTLE RESISTANCE TO FLOW OF AIR.

The fan pipe is of two sizes. The 16 in. pipe is made from 14 gage galvanized ingot iron, and the 11 in. from 16-gage galvanized ingot iron. Both sizes are made in our own shops at a cost of $6.80 and $5.60, respectively, per 8-ft. length.

Flexible tubing has been used but little in the past, although its field of application is being extended and it is now in use wherever conditions permit. Comparative cost figures are not available, but it is safe to say that, for general use, the metal pipe gives much longer service. Installation and transportation costs are in favor of the flexible tubing. Unless the tubing is thoroughly protected from blasting and otherwise properly cared for, its life is short.

Water sprays are used to combat dust, particularly at the loading bins in the Hopewell adit, and at the transfer points and loading chutes of waste raises. The sprays are a combination of compressed air and water, the air tending to break up the water into a very fine spray which throws the dust downward into the cars and, due to the moisture absorbed, it drops out of suspension of its own weight. The spray consists of a 1-in. pipe, having slots cut every 6 in. The pipe is fastened horizontally above the chute with the slots facing downward. This arrangement makes a very effective spray and when properly made uses a minimum of compressed air and water.

PERSONNEL

A ventilation engineer is employed. His duty is to see that the mine is at all times properly ventilated, to see that the air courses are maintained, and to keep all equipment in good repair. In addition, the operating department must be kept informed in a general way of any changes that may be made so that foremen and shift bosses have a good knowledge of the air currents on their particular beat. It is further necessary that the mine fire foreman have a thorough knowledge of the mine and the air currents. He, too, is kept informed as to changes which are made from time to time.

The construction work, such as doors and bulkheads, the installation of small fan units, pipe and tubing, etc., is done by timbermen who work under the direction of the ventilation department.

MAPS AND RECORDS

A set of up-to-date mine maps is kept in the ventilation office. These are 30-scale tracings and show all the necessary ventilation data. Information which is subject to change is marked on the maps in pencil and dated. In addition to this a detailed record of working places is kept on file in a large loose-leaf notebook. This gives the data, working place, psychrometric readings, percentage relative humidity, the amount of air passing, number of men, type of rock, method of mining, the condition of the air entering the stope, and a column for remarks.

INSTRUMENTS AND EQUIPMENT

Equipment consists of a 4 in. anemometer, a sling psychrometer, a small portable water gage, a complete Pitot tube outfit, a recording pressure gage at the fan, and thermometers at important points throughout the mine.

COST OF VENTILATION

Table 6 shows ventilation costs from 1923 to the end of September, 1928.

These figures may appear high, but they include all work chargeable to ventilation. Some of the special items include the downward extension of the air shaft from the 1950 to the 2400-ft. level, and 1100 ft. of return air raises, the fresh air intake shaft, which is 980 ft. deep and 13 by 13 ft. in the clear, the new fan installation and many improvements in the return air course.

TABLE 6.—*Ventilation Costs*, 1923–1928

Year	Total Costs	Cost per Ton
1923	$ 47,785.06	$0.060
1924	90,145.46	0.085
1925	102,834.93	0.107
1926	137,517.17	0.173
1927	115,019.73	0.160
1928 (first 9 months)	73,153.55	0.126

RESULTS

In conducting the ventilation work of the United Verde mine, the aim of the entire organization has been to improve the air conditions at the working face. Since the conditions which require improvement are principally heat, high humidity and occasionally a lowered oxygen content of the air, the remedy is greater air circulation. A comparative study of the air conditions in the stopes over a period of time should show the results of the ventilation work.

Kata readings would no doubt interpret fairly accurately the atmospheric conditions in a working place. The instrument is, however, quite delicate and the fact that hot water must be carried along when taking such readings has been against its use. Psychrometric readings and percentage relative humidity derived therefrom, together with anemometer readings are relied upon as a means of measuring improvements. The disappearance of fungus growths, the fact that men no longer must wait on the station for the smoke to clear out, and a decrease in the number of gas accident cases are all good indications of improvement and increased efficiency.

Table 2 gives the average temperature and relative humidity of certain stopes on various levels of the mine as they existed in the fall of 1925. Unfortunately, records are not available for all the stopes. The temperatures can not be considered high with 81° F. as the maximum and 75° F. as the minimum. The humidities, however, are somewhat different; only 14 per cent. are below 85 per cent. relative humidity, and 63 per cent of the 22 stopes represented are over 90 per cent. relative humidity. These stopes were not selected for their high humidity; as a matter of fact, they were at the lower end of the ascending air current and should have been better than the stopes above them. Table 2 gives the average temperatures and percentage relative humidity of practically all the active stopes by levels in the entire mine, as taken Oct. 15 to 18, 1928. Of the 44 stopes represented only three, or 9 per cent., are 81° F. or over, the maximum being 82° F. Thirty-four stopes, or 77 per cent., are less than 75° F. In the relative humidity column, we find 22 stopes, or 50 per cent. are below 85 per cent. in relative humidity, and only eight stopes, or 9 per cent. are over 90 per cent. in relative humidity. Fourteen stopes, or 32 per cent., are below 70 per cent. in relative humidity. These changes of reduced temperatures and relative humidities have been brought about by a greater circulation of air. The combined effect of these three factors is considered the standard by which the results obtained from ventilation work can be considered.

Table 7.—*Stope Temperatures and Relative Humidities*

[Readings taken Oct. 14 to 17, inclusive, 1927 and 1928]

Level and Stope	Temperatures, Deg. F.			Relative Humidity, Per Cent.		
	1927	1928	Net Change	1927	1928	Net Change
300 16-C	69	65	−4	100	89.5	−10.5
400 16-C	69	63	−6	100	89.3	−10.7
500 15-E	67	64	−3	94.7	69.6	−25.1
600 16-F	84	65	−19	91.4	69.0	−22.4
1200 16-L	72	65	−7	64.8	69.0	+4.2
1650 Top Slice	74	65	−9	81.8	69.0	−12.8
1800 11-Q	73	68	−5	90.1	85.4	−4.7
2100 16-S	82	76	−6	96.0	86.6	−9.4
2250 7-O	77	68	−9	95.7	85.4	−10.3
9-R	74	73	−1	90.4	95.3	+5.1
11-T	77	73	−4	90.7	65.2	−25.5
12-T	78	75	−3	90.8	54.0	−36.8
2400 12-U	77	73	−4	78.4	46.8	−31.6
14-U	77	76	−1	78.4	44.2	−34.2

New fan put into operation Oct. 23, 1927.
Return air course opened to 2100-ft. level May 2, 1928.
Upper level ventilation started (500-ft. level fan) May 8, 1928.

Table 7 shows some important changes. One group of readings in the 1927 columns was taken Oct. 14 to 17 of that year, which was a few days prior to the date on which the new fan was cut in (Oct. 23, 1927). The other series of readings was taken one year later, or after the new fan had been in operation for about one year. The return air course was opened in May of that year and the air splits were also arranged at about the same time. The results obtained from these changes in the ventilation system speak for themselves, and need not be enlarged upon. They alone, without any other benefits, justify the expenditure which was necessary for 600 ft. of raising for the downward extension of the return airway to the 2100-ft. level and the installation of the new fan.

Table 8 gives some indication of what better ventilation can do to help the mine safety record. The first line indicates the gas cases which involved the loss of one shift or more; the second line indicates those cases which necessitated laying off for the remainder of the shift only, being so slight that the men were able to return to work the following day.

TABLE 8.—*Gas Cases,* 1924–1928

Cases	1924	1925	1926	1927	1928
Lost time	6	15	7*	5	
Slight	8	6	8	3	6
Total	14	21	15	8	6

* Includes two fatalities.

VENTILATION AND MINE FIRES

At the United Verde, ventilation work is not only carried on for the general improvement of working conditions, but also with a view to limiting the possible hazards of mine fires. I have previously made the statement, and believe that it bears repeating, "When a fire occurs, it is the ventilation system that will save or destroy life, depending on how good the system is, how thoroughly it is understood, and how well emergency problems have been worked out in advance."

A mine fire foreman is in charge of all mine fire work as well as the prevention and fighting of mine fires. (Fig. 12.) This work and that of ventilation are inseparable in many respects.

Horizontal fire breaks have been established on the 1950, 1650 and 1200-ft. levels. These consist of level pillars left in place in the course of mining, and fireproof coverings over raises coming through the level pillars. A reinforced concrete collar is placed in each of these raises, and an iron trapdoor, hinged to the concrete collar, is used as the cover. The trap doors are kept open or closed, depending on ventilation requirements. Thus, on the 1950-ft. level, they are kept closed, which makes possible the elimination of air on the 2100-ft. level. In case of a fire below the 1950-ft. level, the fire would be prevented from moving upwards past the level by the fire break at that horizon. All doors in raises being closed, the smoke would be diverted to the return on the 2100-ft. level, thus permitting operations to continue above the 1950-ft. level fire break. The fire breaks on the 1650 and 1200-ft. levels function in a similar manner.

As shown in Fig. 13, the air shaft is so situated that as long as some fan pressure is maintained, it is practically impossible for smoke from a fire to reach shaft 6, through which the men are hoisted. To keep the fan running during a fire would mean a forced draft for the flames, and to overcome this difficulty certain doors in the mine (one in the main drift of each level) are painted red and designated as "fire doors" (Fig. 14). On these doors is stenciled in large letters, in both English and Spanish, "In case of fire close this door after all men have passed through to No. 6 shaft station." The shift boss is expected to check the men out and close the door. When these instructions are followed, the entire ore zone or stoping area will be cut off from fresh air immediately after all men have passed on their way to the shaft.

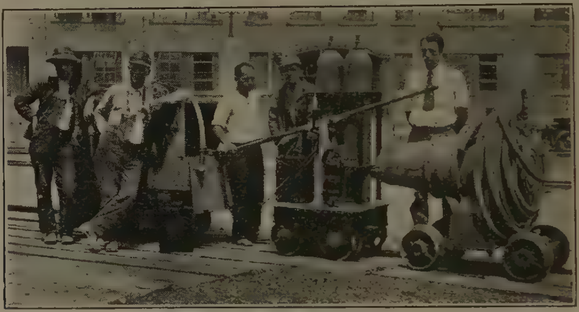

FIG. 12.—MOUNTED FIRE-FIGHTING EQUIPMENT; FIRE TOOL TRUCK, OXYGEN PUMP AND TANK TRUCK AND A No. SP350 COPPUS VANO BLOWER.

The fire signal is nine flashes of the light circuit given three times and followed by the level signal to indicate the location of the fire. When this signal is flashed through the mine, the fan speed is reduced to slow speed, and a door near the fan station is opened. This establishes a fresh air split direct from the fan to shaft 6, precluding any possibility of smoke filling the main exit. In the meantime, the men are leaving their working places and closing the fire door behind them. There will then be no forced draft to feed the flames. The shaft and a part of the main drift on each level will be in fresh air. Shaft 6, which is the main exit, will be on an independent fresh air split direct from the fan and will be supplied with 80,000 cu. ft. of fresh air per minute.

Reconnaissance of the district on fire and the actual fire fighting should be much easier under the conditions outlined because a fresh air base can be established immediately on any level and within a short distance of the fire.

The fire signal is given once every three months for practice purposes. The ventilation system is arranged in these drills as outlined above. It requires from 6 to 8 min. to get all men on all the levels to the shaft station.

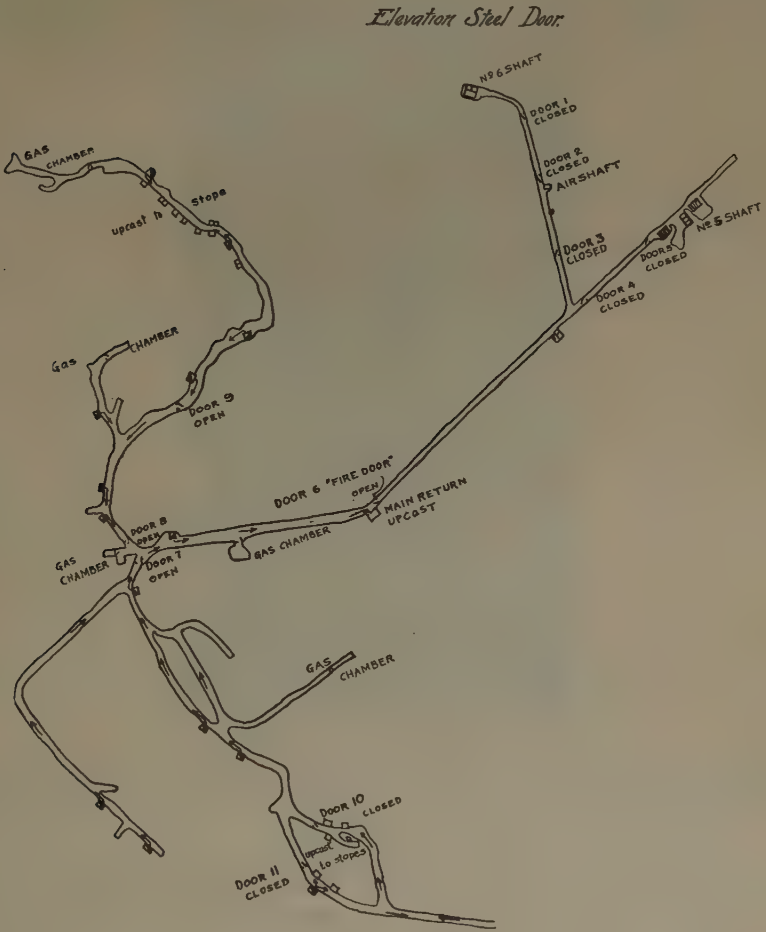

FIG. 13.—PLAN OF 2100-FT. LEVEL SHOWING VENTILATION SYSTEM.

No condition is known at this time that might arise under which the fan would have to be reversed. However, to safeguard against the unknown, the fan is reversed once every two months to see that all parts are in proper working order. A report is made in writing of this test to the general mine superintendent, the mine superintendent, and the chief engineer.

SAFETY CHAMBERS

Dead ends of drifts in the far ends of the levels are used as safety chambers. A door is installed in such a drift as far from the face as conditions permit, so as to give as large a space inside as possible. (Fig. 15.) These chambers are supplied with electric lights, compressed air and water lines. A sign on each safety chamber door states that the door should be closed and the air turned on in case of emergency use. Thirty-five safety chambers have been equipped, each level having two or more.

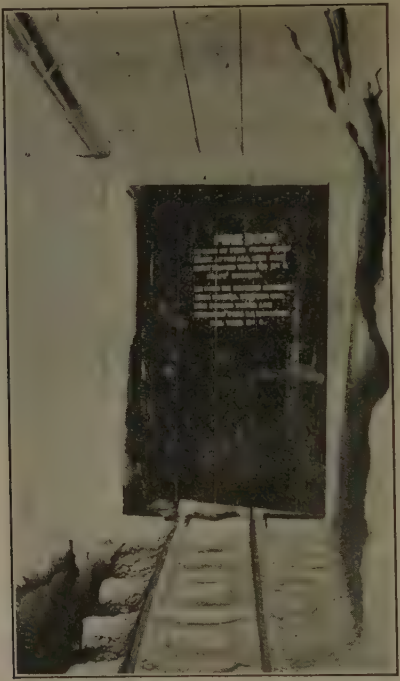

FIG. 14.—A "FIRE DOOR."

Door is painted bright red and is closed in case of fire, after all men have gone to shaft station.

DUAL POWER SUPPLY

The main electric feed lines for the two hoists and the fan pass through the 500-ft. adit. To safeguard against the possibility of having the power cut off indefinitely because of a fire or some other trouble in the 500-ft. adit, a second power line will be installed through the Hopewell adit on the 1000-ft. level.

Acknowledgments

The writer is indebted to W. V. DeCamp, general superintendent and to W. W. Lynch, mine superintendent of the United Verde Copper Co., also to D. Harrington, chief engineer of the Safety Division of the U. S. Bureau of Mines, for their kind assistance and helpful criticism in the preparation of this paper; also to C. E. Mills, chief engineer of the United Verde Copper Co., for the use of valuable data and drawings.

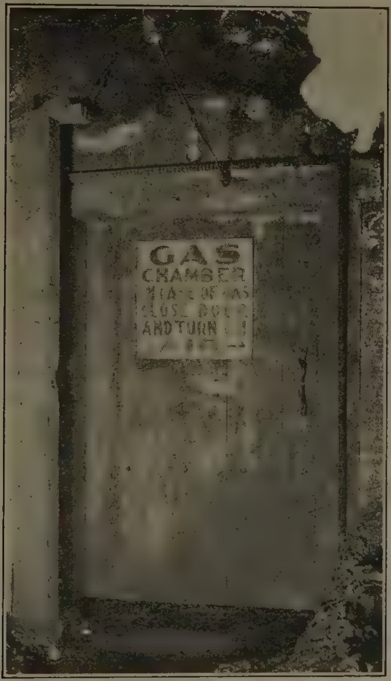

Fig. 15.—Outside view of a "gas chamber" door.
Note notched compressed air pipe around door frame

DISCUSSION

D. Harrington, Washington, D. C. (written discussion).—This paper describes so many unusual and generally excellent practices in connection with ventilating a metal mine that the various briefly outlined activities might well be extended to cover a rather voluminous book. At any rate, it seems that it would be decidedly desirable to have detailed papers on the following phases of ventilation or fire-prevention practice touched upon in this paper:

1. The construction and operation of the injector-type blower mentioned on page 132.
2. The construction, placement and operation of the 400 or more doors in the United Verde mine.

3. The system of splits in the ventilation of the United Verde mine (an abnormal practice in connection with metal mines, though comparatively common in coal mining).

4. The method of providing horizontal fire breaks in the United Verde mine (also abnormal for any kind of mine, coal or metal).

5. The establishment and maintenance of underground safety chambers to the number of 35 or more (an abnormal practice and one which has been of immense benefit in this mine).

6. The blasting practice, which is well worth an extended paper, involving as it does the absolute necessity for preventing noxious blasting fumes from being breathed by workers.

A reading of the paper convinces one immediately of the definite desire of the United Verde Copper Co. to avoid any possibility of having the air of the main intake air course of the mine vitiated by possible fire fumes; this can be noted on practically every page of the paper by Mr. Glaeser. There is also to be noted a very definite desire on the part of the company to safeguard the employees from fire or other fumes at all times and in all parts of the mine.

To those who are interested in the ventilation of metal mines, one of the novel features brought out by this paper is the excellent system of air splitting and the continued effort on the part of the company to provide separate sections of the mine with their own air flow and to keep that air from flowing through other sections of the mine after it has ventilated the one section to which it was originally diverted. This practice is found in few if any other metal mines in the United States.

For the protection of the employees of the mine at or after a possible mine fire, the main hoisting shafts are so surrounded by doors, ventilation currents, etc., that it is next to impossible to have these important adjuncts of the mine at any time filled with smoke and hence nonaccessible to those who wish to leave the mine. The use of doors so placed that any shaft may be separated from the mine, from other shafts, or from almost any part of the mine is an excellent practice, which should be employed in all mines, coal or metal; however, it is rarely found in either. The fact that the mine has placed more than 400 doors indicates that the company realizes the extreme desirability of being able to isolate sections of the mine as well as definite splits in case of fire. Moreover, this multiplicity of doors is of use not only at time of fire but also in the coursing of air at ordinary periods; one of the interesting features of the report is the description of the practice of preventing any air except the main intake air from going *downward*. In other words, after the fresh air is released from the intake shaft, the air is from that time forced to proceed upward in all parts of the mine, and at no time or place in the mine is air allowed to go down after it has once started on its way from the main intake shaft. Such control is of vital importance at time of fire and is of scarcely less importance at time of blasting in a mine which has dangerous gases such as are found in the United Verde; at blasting time these gases are SO_2, H_2S, and CO.

Another exceptional feature brought out in the paper is that of providing definite return air courses for a metal mine; also the excellent practice of removing timber from raises, or stripping them, is one that could very well be followed by a considerable number of metal mines in order to increase the opportunity for the escape of return air. The paper also brings out the fact that the distribution of the large quantity of intake air to the various working faces is by no means neglected, and that although the mine uses several decidedly different kinds of methods for underground extraction of ore, provision is made in all working places for circulation of air; in this mine the driving of raises is kept well ahead of stope work in order that the stopes can be kept well ventilated. The mine also uses overcasts, another innovation rarely met in a metal mine.

The dust of this mine is an oddity in metal-mine dusts in that its high sulfur content renders it explosive, at least in some places. In the United Verde mine, sulfur in some instances constitutes over 40 per cent. of the material mined. The paper brings out the fact that while the ventilation system is such that it is almost impossible to conceive of any reason why it would be necessary to reverse the direction of the air flow, nevertheless provision has been made so that the air flow can be reversed if necessary; moreover, this feature is at times tried out to make certain the air flow can be reversed without undue delay if desired.

The keeping of an attendant at the fan at all times is an extraordinary precaution seldom found even at coal mines, and almost never at metal mines. The placing of not only one, but several employees on full-time work in connection with ventilation is another oddity which is rarely found in either coal or metal mines. The costs in connection with ventilation, as shown on page 135, indicate that this metal mine is expending not only more attention and time, but also money, in the establishment and maintenance of good ventilation that can be found in almost any coal mine in the United States.

The tabulation on page 117 showing the temperature and humidity of intake air and, giving the temperature of the air at the surface as well as at various points along the intake airshaft from the surface to the 2400-ft. level, proves beyond a doubt the fallacy of the theory that underground air temperatures are determined wholly or even largely by adiabatic compression. Here the data for Aug. 20, 1928, when the surface air temperature was 92°, shows that instead of there being an increase in temperature as the air proceeded to a point 2400 ft. vertically down into this mine there was an actual decrease of 10°, from 92° F. to 82° F. There is absolutely no question that underground air temperatures are determined by rock or water temperature largely if not almost wholly, and that while adiabatic compression undoubtedly enters into the matter to some extent, it does not by any means determine the temperatures of underground workings.

The utilization of the air coursing to aid in the prevention of decay of timber, as outlined in a few words on page 119, is of decided importance. There is no question that the United Verde Copper Co. has been able to save a considerable amount of its ventilation expenditure by the greatly decreased amount of timber decay which it has been able to effect by manipulation of the air. The atmospheric standards, as indicated on pages 115 and 123, are interesting and rather drastic, yet it is entirely probable that the maintenance of these standards will pay well in the long run not only in health and safety but also in dollars and cents. In a mine where as many precautionary measures are in effect as at the United Verde the use of closed lights would be expected, or to put it differently, the elimination of open lights, and there is no reason to doubt that sooner or later closed lights will be adopted. Moreover, it seems definitely advisable that ventilation, fire prevention, and fire protection work, as well as safety in general, should all be under the direct supervision of some one person; this, too, undoubtedly will be arrived at sooner or later.

In general, this rather short pamphlet discusses so many excellent practices that it is unfortunate that it could not have been expanded to several times its present size in order to give more of the details than were possible in a short paper.

Mine Development and Underground Construction of Andes Copper Mining Co. at Potrerillos, Chile

By I. L. Greninger,* Potrerillos, Chile

(New York Meeting, February, 1929)

The property of the Andes Copper Mining Co. is in the central part of the Republic of Chile, Province of Atacama, 147 km. (91.5 miles) eastward by rail from the port of Chañaral. At this point the Andean system is made up of two principal ranges. The mine is on the westerly slope of the outer, or western, range, at a mean elevation of 3250 m. (10,500 ft.).

A meter-gage railroad extends from the company's port at Barquito, near Chañaral, to the reduction plant at Potrerillos. This road has a total length of 151 km. (94.0 miles), of which 93 km. (57.79 miles) are owned by the Andes Copper Mining Co. The remainder of the road is owned by the state. The company operates its trains over the portion of the road owned by the state, under an agreement with the Chilean Government.

The mine is connected with the reduction plant by a meter-gage railroad, 10 km. (6.214 miles) long, which extends from the coarse-ore bins at the metallurgical plant to the underground storage bins at the mine. Tunnels make up 60 per cent. of the total length of this railroad.

As is characteristic of the Andean region in Chile, the topography of the locality is extremely rough, and a number of points near the mine have elevations of from 15,000 to 16,000 feet.

History of Potrerillos Mine

The ore occurrence has been known for many years, and many of the claims under individual ownership were worked in a small way before the property was acquired by the present owner—the Andes Copper Mining Company.

The highest grade oxide ores near the surface were mined by primitive methods and, after sorting, the product, assaying around 15 per cent. copper, was hauled by mule carts a distance of about 80 km. (49.71 miles) to Pueblo Hundido on the State Railway System, shipped to the Port of Chañaral by train and thence to Europe by water.

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*Mine Superintendent, Andes Copper Mining Co.

The early history of the Potrerillos mine, as disclosed by public records of the district, is as follows:

In the year 1894 Don Felipe Tapia denounced two mining claims in what is now the Potrerillos district, which he called the "San Antonio" and the "Quebradita." This appears to be the earliest official record covering any part of the property now owned by the Andes Copper Mining Co. In 1896 Don Eduardo Tellez entered the district as owner of a number of claims. In the same year Tellez, together with Echeverría Cazote, Echeverría Larraín and others, organized a company known as the Compañia Minera de Potrerillos. This company acquired at the time practically all the claims recorded in the district to that date. In 1913 William Braden acquired title to all the mining claims of the Compañia Minera de Potrerillos and denounced many additional claims. In 1916 Mr. Braden sold his entire interest to the Andes Copper Mining Co. At present the company owns 294 metal-mining claims containing a total of 1430 ha. (3532 acres). These are recorded as being in the "Agua Dulce" mining district.

GEOLOGY OF THE ORE DEPOSIT

The ore is found in an intrusion of porphyry, generally classified as quartz-diorite porphyry. This has been forced upward through thinly bedded sedimentaries, and the sedimentary strata have been broken and tilted as a result of the upward movement of the porphyry.

The surface area of the intrusion measures about 2 km. from north to south and has a maximum width of 1 km. (0.6214 mile). Generally, the width is greatest near the northerly limit, and the intrusion, having a rough wedge shape, tapers to a point at the extreme southerly end.

All of the porphyry contains some copper, but a considerable portion of the intrusion is of too low a tenor to be considered as ore.

The porphyry intrusion has left the sedimentaries—quartzite, limestone and sandstone—tilted at an angle of from 50° to 60° from the horizontal, and the orebodies within the intrusion generally have the same dip as the sedimentaries.

Fig. 1 shows the general shape of the intrusion and the outline of known ore. The copper is found in the form of malachite, azurite, chalcocite and chalcopyrite, with many other of the less common copper minerals in evidence. Of the total tonnage developed, about 35 per cent. is in the form of oxides and 65 per cent. in the sulfide form.

Fortunately, there is a very small tonnage of mixed ore, the change from oxide to sulfide being sharp. At most points there is a thin blanket of leached porphyry between the oxide and sulfide ores, and the upper portions of the sulfide measures show the effect of this leaching through secondary enrichment in sulfide mineral. The zone of secondary enrichment is generally not more than 20 m. thick.

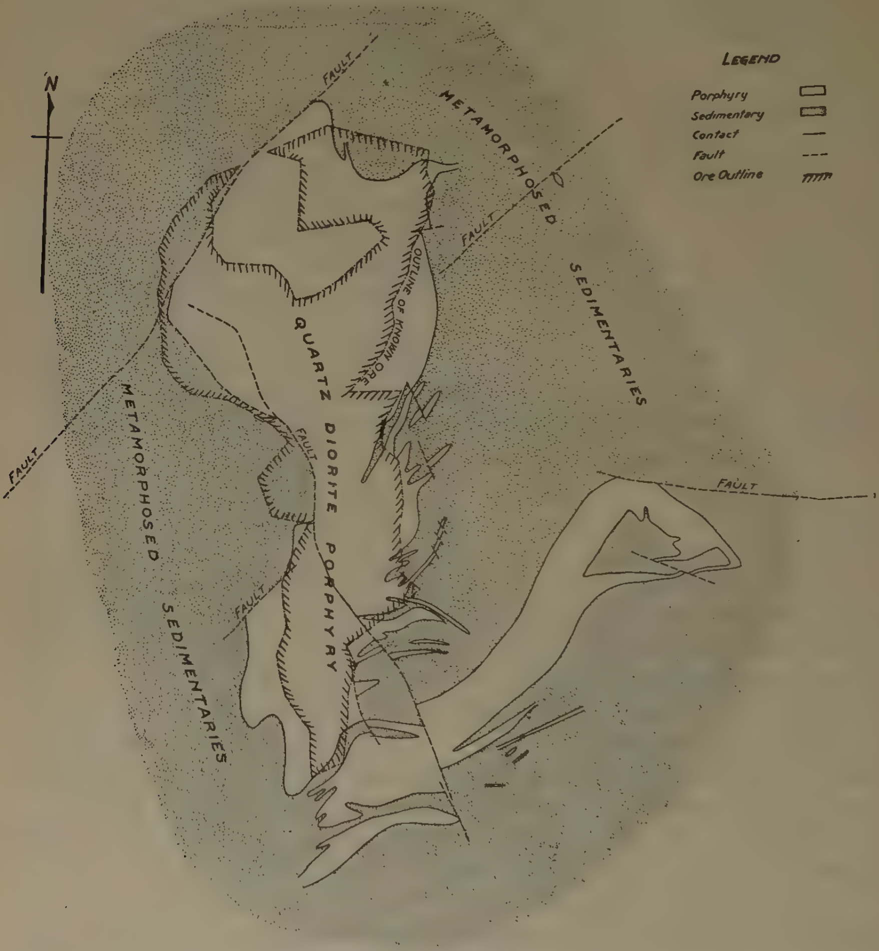

FIG. 1.—GEOLOGICAL PLAN SHOWING OUTCROP OF PORPHYRY INTRUSION AND OUTLINE OF KNOWN ORE.

EXPLORATION

When the present owners became interested in the property, a campaign of churn drilling was inaugurated and was continued for a number of years; 168 holes were put down. These holes varied in depth from about 50 m. (164 ft.) to 388 m. (1273 ft.)—the greatest depth attained in any hole. The ground is soft and well suited to exploration by this method, except that caving of the holes was common, thus making necessary frequent changes in size of casing and reducing possible depth of holes. The total length of holes drilled amounted to 28.80 km. (17.90 miles).

Prior to the year 1918, exploratory openings in the form of drifts and raises were driven to a total of 11,216.61 m. (36,790.48 ft.). This work, together with the churn drilling, served to prove the general location and form of the ore measures, and a plan of general development was decided upon early in the year 1917.

General Mining Plan

According to the plan adopted, an adit was to be driven under the then known ore. Driving of the adit was started in May, 1917, and the tunnel was completed to location of first ore bin in 1920. The general layout was to consist of:

Main adit, at elevation of 2974 m. (9754 ft.), 2.17 miles long;

Storage bins, from which ore could be loaded into cars for delivery to reduction plant;

Intermediate haulage level—elevation 3184 m. (10,443 ft.);

Ore passes connecting ore bins with intermediate haulage level;

Mining of sulfide and oxide ore above intermediate haulage level;

Driving of a second level at a point to be later determined, in order to serve for mining of sulfide ore below intermediate haulage level.

This general plan has been adhered to, with such additions and changes as have seemed advisable from time to time.

The undercut caving system (sometimes called "block caving") was adopted both for mining ore and removal of waste. This system was first used in the United States in connection with non-ferrous metal mining at the property of the Ohio Copper Co. in Utah and later was adopted by the Inspiration Consolidated Copper Co., at Inspiration, Ariz. It has since been used, with certain modifications, by the Nevada Consolidated at the Ruth mine, the Miami Copper Co., and others. Credit for pioneer work in connection with this system is largely due to Felix MacDonald, who is now, and has been for many years, mine superintendent for the Inspiration Consolidated Copper Co. at Inspiration.

Exploration by churn drilling showed that at some points the oxide ore reached the surface; at other points there was barren capping. It was determined that on some portions of the orebody the capping was comparatively shallow and could be profitably removed, while in other localities the overburden was too thick to permit of its removal. Generally the northerly part of the orebody has little or no capping, while the southern sulfide orebody is covered to a vertical depth ranging from

100 to 200 m. (328 to 656 ft.). Metallurgical convenience dictated that mining be started first on the sulfide ore, and it was therefore decided to begin work at the south end of the orebody, as mining could be started there without waiting for removal of waste or oxide ore. As a matter of fact, there were several million tons of oxide ore in this locality which, while not directly overlying the sulfide ore to be mined first, was within the surface outline of the cave that would result as mining operations progressed. It was found that this oxide ore could be mined and stored for future recovery, in time to be out of the way before the cave resulting from the mining of sulfide ore endangered it.

Removal and Storage of Ore from South Oxide Orebody

Preparation for mining and storage of the oxide ore was started in April, 1925. Actual removal of ore was first accomplished in April, 1926, and to date 3,000,000 tons have been removed and stored at a point where it can be readily reclaimed when needed. An additional 2,000,000 to 2,500,000 tons remains to be taken off from the region which will be endangered by caving of sulfide ore at a lower elevation.

In mining this oxide ore, the regular caving method is used, except that no control is practiced, as no capping exists over the ore. No timbering of any kind was put in the branch raises, and ore was drawn from any raise that happened to be open. In this work, haulage or loading drifts were driven at intervals of 30 m. (98.40 ft.). Grizzly drifts were spaced at 15 m. (49.20 ft.) and were 12 m. (39.36 ft.) above the haulage level. Undercutting drifts were driven on 7.5-m. (24.60 ft.) centers, and 10 m. (32.80 ft.) above grizzly level. Two branch raises connected with each grizzly and these raises in turn had four branches each, connected with the undercutting level. Grizzlies were spaced 11.50 m. (37.72 ft.) center to center.

Fig. 2 is a representative section through the South Oxide orebody, showing the outline of the deposit and arrangement of openings driven for handling the ore. Haulage drifts, in section, are 8 ft. high by 8 ft. 6 in. wide. Chutes are equipped with hand-operated arc gates. Cars are of the rocker dump type and have a capacity of 2.25 cu. m. (79.44 cu. ft.). They are equipped with roller bearings and are handled by 10-ton trolley-type locomotives taking current at 250 volts. Trains are made up of 24 to 32 cars each.

In connection with disposal of this ore on surface, great care had to be used in handling and moving the dump track. The vertical height of the dump is now about 60 m. (196 ft.) and heavy slides have happened at times, which have carried away portions of the track. In order to guard against this danger, the track is anchored by means of cables which are attached to "dead men" buried in the dump. The cables are slackened when it becomes necessary to move the track but are always kept tight when the track is in use.

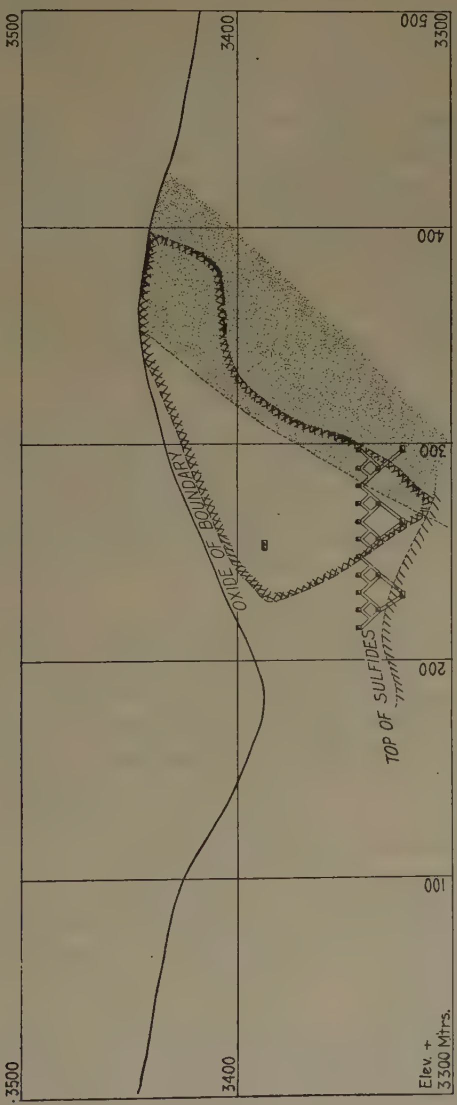

Fig. 2.—Representative transverse section, south oxide orebody, showing haulage drifts, main and branch raises.

Waste Removal

Waste capping is also being removed at a number of points over the north orebody where the mining practice is exactly like that described above for handling oxide ore. Openings are of the same dimensions and have the same spacing with the exception that the undercutting horizon is made to conform as closely as possible to the top of ore.

A deposit of alluvial gravel was encountered, filling a deep canyon which traverses the central part of the orebody. It was impossible to handle this material by the usual caving methods and our system was therefore modified to suit the material involved. From a haulage drift driven below this alluvial deposit a number of raises were driven through the ore. Some of these raises were extended to the surface and were enlarged by using the "mill-hole" system, until the material around each raise had assumed the angle of repose. We then used a Bagley grader. This device was handled by means of a double-drum hoist having a rope pull of 20,000 lb. The gravel, which was lightly cemented, was loosened by blasting with black powder and was then scraped into the various raises with the Bagley grader. This system proved to be highly efficient and the gravel was transported to the raises and trammed to the dump at a total cost of about 23½ c. per short ton.

The entire mine townsite, as it now exists, is built on ground made by the removal of waste. The old mine camp was built in a steep canyon and the character of the surrounding country did not admit of expansion, therefore it was decided to use the ground made by the disposal of waste for the new townsite, and all buildings constructed since 1925 have been built on this filled area.

Mining of Sulfide Ore

Preparation for mining ore in the south sulfide orebody was started in June, 1925. In December, 1926, ore production was started. Fig. 3 represents a transverse section through the orebody where mining was initiated and also shows the workings on and above the intermediate haulage level.

Fig. 4 is a plan of the intermediate haulage level showing spacing of drifts. Haulage drifts are driven at intervals of 33.33 m. (109.32 ft.) and are turned off from the gathering drifts at an angle of 42°. Main raises are located to conform to car lengths and are spaced at 6.25 (20.50 ft.) and 9.37 m. (30.73 ft.). The grizzly drifts are spaced 16.66 m. (54.64 ft.) from center to center, and located 12 m. (39.36 ft.) above the haulage drifts. Branch raises are driven at intervals of 8.33 m. (27.32 ft.) along the grizzly drifts, and the block served by each branch raise system measures 8.33 by 8.33 m. (27.32 by 27.32 ft.). Undercutting drifts are spaced 8.33 m. (27.32 ft.) center to center and 10 m. (32.8 ft.) above the grizzly level. The block first mined extended entirely across the orebody from west to east, a distance of 200 m. (656 ft.) and measured 70 m. (229.60 ft.) from north to south. Experience has demonstrated that this block was somewhat larger than was advisable and new blocks are being laid out, extending across the ore measure but having a length from north to south of only 50 m. (164 ft.).

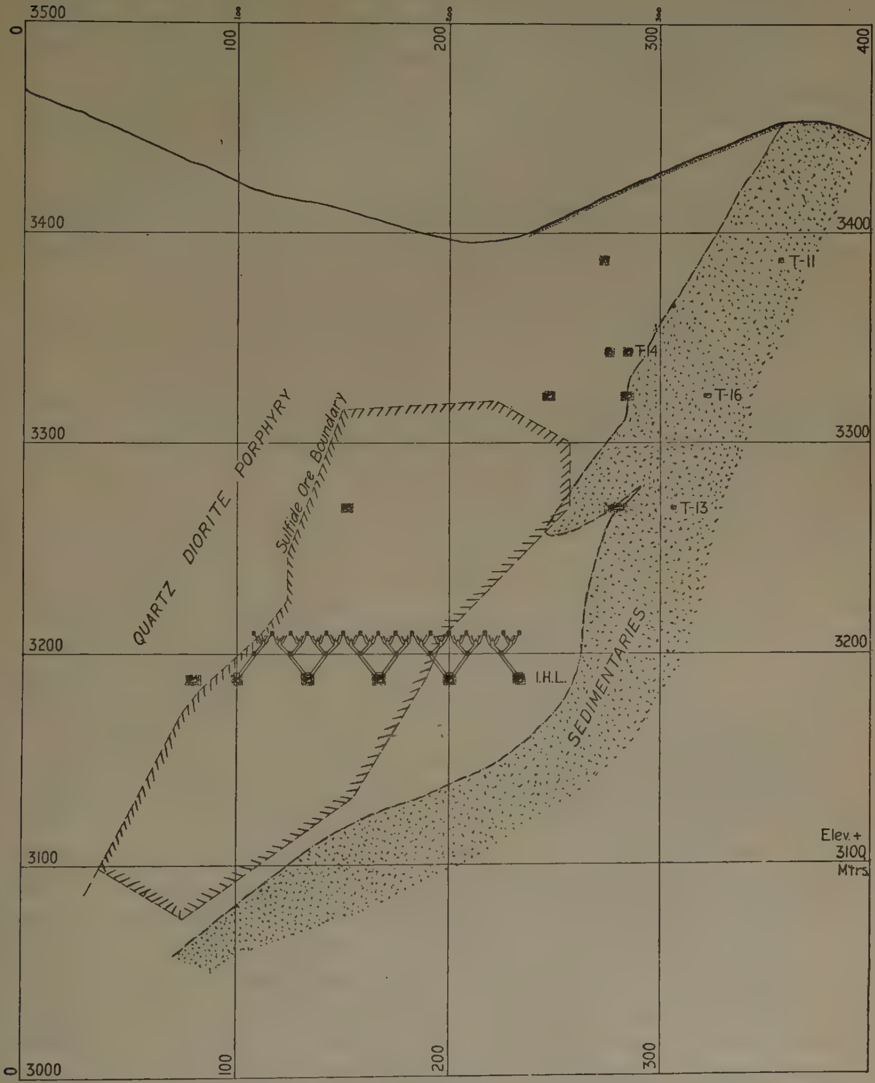

Fig. 3.—Representative transverse section, south sulfide orebody, showing haulage drifts, main and branch raises.

In each branch—or drawing-off—raise, there is constructed a control set having four chutes. Branches are driven upward from these chutes and connect with the undercutting level at fixed intervals. Fig. 5 is a plan and section of branch raise system showing the spacing of branch raises. Chute openings in control sets measure 3 by 3 ft. and have ordinary wooden gates. Angle iron 3 by 3 by ⅜ in. is used for gate supports, as this material has proved much more satisfactory than wood.

Undercutting is accomplished by driving a short crosscut from a given undercutting drift where work is to be started. The crosscut is made use of in order to provide room for handling long steel and to avoid expense of supporting a large opening where ordinary widening of the drift is practised. The crosscut is driven into the pillar a distance of 4.5 m. (14.76 ft.) and from this cross-cut undercutting holes 3 to 4 m. long (9.84 to 13.12 ft.) are drilled. Undercutting advances diagonally across the block and, in case of Block 1, was started at the westerly limit and carried across to the eastern side of block, where it was completed.

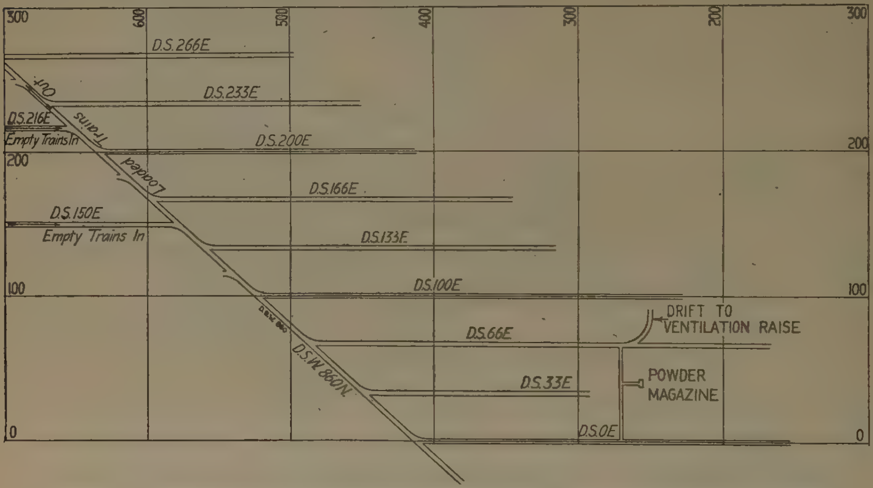

FIG. 4.—PART OF INTERMEDIATE HAULAGE LEVEL, SOUTH SULFIDE OREBODY.

Because the orebody dips at an angle of about 50°, it is necessary that a large proportion of the grizzly and undercutting openings be driven in the footwall. As the formation here is a badly broken sedimentary, crushing and closing of grizzly drifts and branch raises have caused much trouble. In order to avoid this difficulty, recently the grizzly drifts have been lined with rubble masonry; the lining having an elliptical section. The major dimension of the ellipse is 7 ft. and the minor is 4.5 ft. Limestone quarried on the surface is used for this purpose; it is placed in cement mortar. We are also experimenting with reinforced concrete for supporting grizzly drifts. So far the system (using masonry or concrete) has proved highly satisfactory, and although it is expensive, it is less so than the continual repairing of timbers in the grizzly drifts. We have also found that if the grizzly drifts can be supported and prevented from crushing, very little trouble is caused by superimposed weight in branch raises and control sets.

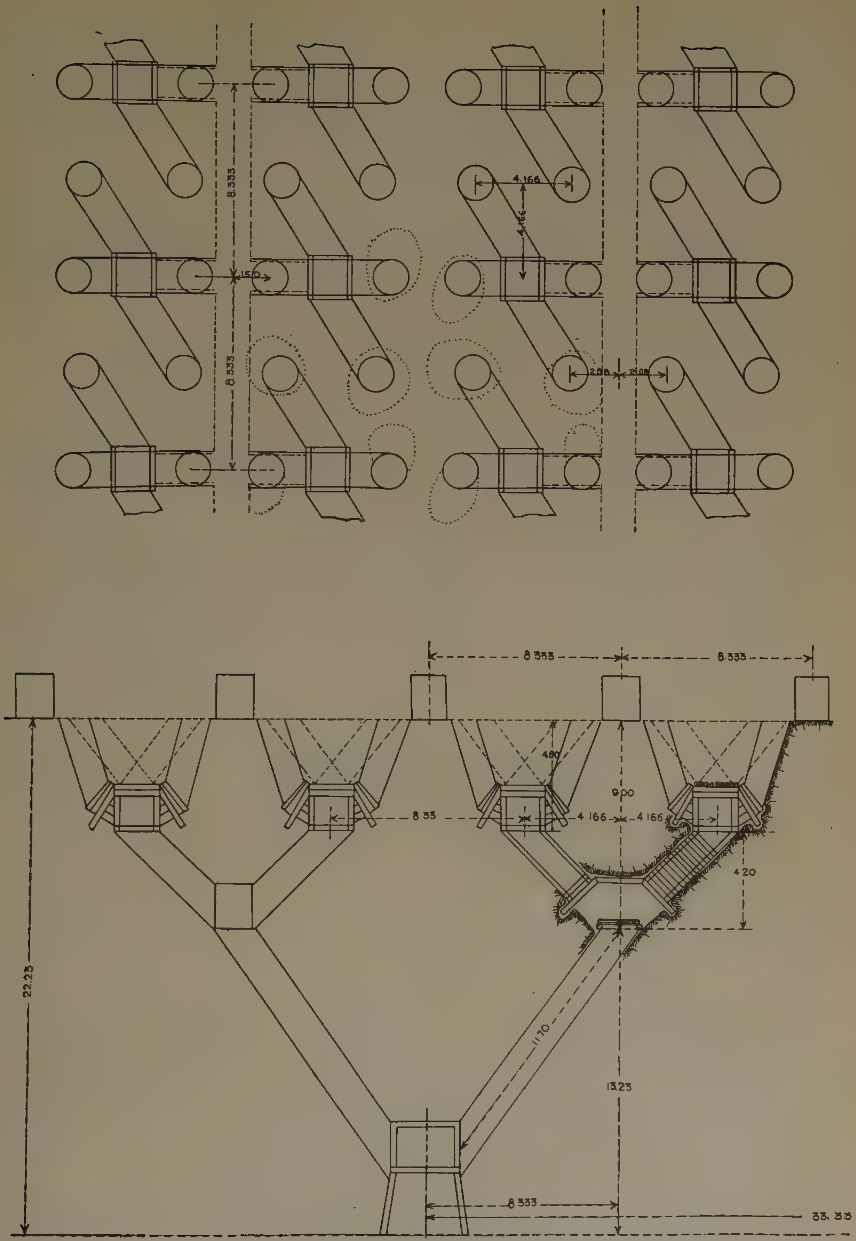

Fig. 5.—Transverse section on main and branch raise system. Plan of undercutting level showing top of branch raises.

TIMBERING

Haulage drifts are supported by 10 by 10-in. Douglas fir. Timber sets are placed 5 ft. center to center, except where chutes are constructed. Chute sets are spaced 3 ft. apart, and as 3-in. plank is used for building chutes, the inside width of chute is 30 in. All timbering is standardized, framing and cutting being done on the surface.

Where a connection is made between haulage drifts, 8 by 10-in. H-beam steel is used for caps, as the span is too long to permit the use of timber where the ground is heavy; and where the ground is exceptionally heavy, 10 by 15-in. steel I-beams are used for caps. Practically all the main or transfer raises must be timbered, as the ground is broken and unstable. In the raises, 6 by 12-in. cribbing is used; the section of raise is 3 ft. 6 in. by 3 ft. 6 in. Cribbing is placed skin to skin and each set is protected with four pieces of angle steel, 3 by 3 by ⅜ in., nailed to the upper corner of the 6 by 12-in. timber. For fastening angle steel, a ⅜ by 4-in. chisel-point spike is used. These nails are much better than the ordinary wire spike for this purpose, as they are annealed and have a countersunk head.

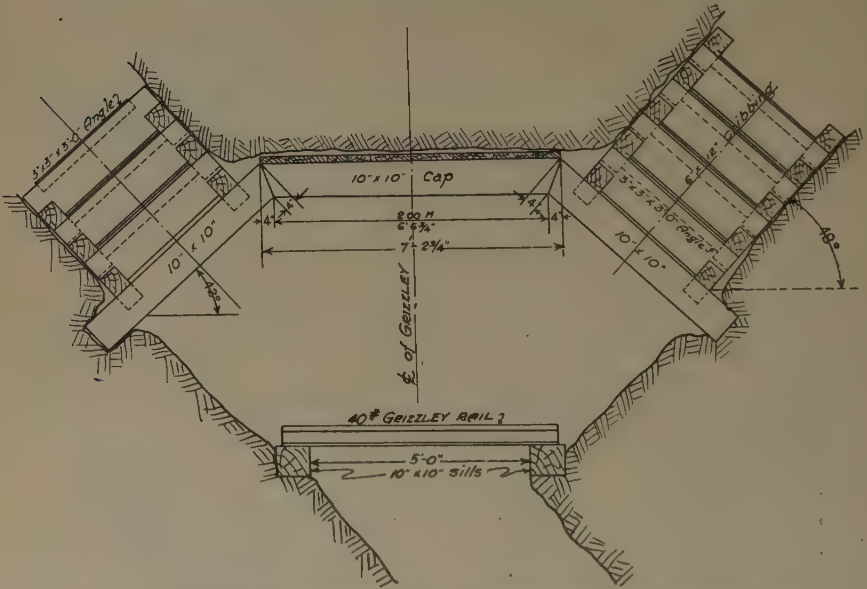

FIG. 6.—ARRANGEMENT OF TIMBERING OVER GRIZZLY, SHOWING SEGMENT SET AND CRIBBING.

It is also necessary that the branch raises above the grizzly level be closely timbered. The same system is used in these raises as for the transfer raises; that is, 6 by 12-in. timber cribbing placed skin to skin and protected with angle steel. A three-piece segment set is used over the grizzly and the posts of the segment set are placed at the proper angle and used as bearers for the cribbing above the grizzly. Fig. 6 shows the arrangement of timbering over the grizzly.

As mining progresses toward the south, the bottom of the orebody becomes progressively higher above the haulage level and transfer raises must be longer. These raises are being timbered with great care, as the loss of only a few sets would result in caving in and closing of the raise. Where branches take off from main or transfer raises serving more than one grizzly, the sets are protected by 8 by 8-in. H-beams to prevent cutting out of the timbers. As noted before, grizzly drifts are supported by masonry lining where there is any danger of crushing. Undercutting drifts are not usually timbered when driven. When undercutting is started and caving begins, it is sometimes necessary that timbers be placed in undercutting openings to support the ground until undercutting rounds have been drilled and blasted.

Ore Drawing

Drawing of ore is under the direction of the Geological Department. Typewritten orders are issued by the chief geologist, with the approval of the mine superintendent, covering ore drawing. At each grizzly two small blackboards are placed, one for each branch raise connecting with the grizzly. On these boards the tonnage to be drawn from each raise in a given week is posted; also the grade of ore in each branch. Grades are brought up to date at least twice each week. Ore drawn from branch raises is checked both on grizzly and haulage levels. A chute checker on the grizzly level observes and notes the raise and branch from which the drawing is taking place. Main raises are calibrated and successively filled and drawn off. An additional chute checker on the haulage level notes the number of cars drawn from each main or transfer raise. The data, assembled by the two checkers, are later compared and checked in the office, and the tonnage drawn from various raises is posted daily on stope sheets.

All bosses having to do with the drawing of ore carry with them a copy of the graphic chute charts on lefax size paper. Graphic chute charts are also maintained in mine foreman's and chief geologist's offices, and all the charts are brought up to date at least once a week. It is to be noted, however, that the work of drawing ore is a continual compromise between opposing requirements. It may happen that a chute is marked overdrawn by the geologist, but if the square set is taking weight it must be relieved in order to avoid excessive repairs. Therefore it becomes necessary to draw a limited amount daily, even from the chutes that may be overdrawn.

The block from which most of the ore has been produced so far was calculated to contain 2,858,588 short tons of ore. From this block we have extracted approximately 2,534,600 tons having an average assay value equal to 90.69 per cent. of calculated grade of ore in place. At present (April, 1928) this block is producing about 165,300 short tons of ore per month.

HAULAGE SYSTEM

For haulage, a Granby-type car with capacity of 5.20 short tons is used. Trains consist of from 16 to 20 cars each. The Granby-type car was developed first by the Granby Mining & Smelting Co. at Grand Forks, B. C., and has since been introduced in a number of metal mines in the United States; for example, the United Verde at Jerome, Ariz. and the Portland Mining Co. at Cripple Creek, Colo. Essentially, this is a side-dumping car which is dumped by means of a block that raises one side of the car as the train passes through the dumping station. Trains of 20 cars can be dumped in about 50 sec., as it is an automatic operation and trains pass through the dumping station at a rate of 1½ to 2 miles per hour. This type of car has proved entirely satisfactory. Trains are drawn by 10-ton trolley-type locomotives using direct current at 250 volts. Total length of the round trip from stope to dumping station averages 2 km. (1.24 miles). Using nine trains, we have handled up to 13,224 short tons in a shift of 8 hours.

The haulage system is so arranged that all principal tracks are used in one direction only, the empty trains returning by a separate route from that used by loaded trains going out.

ORE-PASS SYSTEM

Ore produced above the intermediate haulage level is delivered to a system of ore passes connecting with storage bins on the main adit level. There are three of these ore passes, with six additional passes under construction. The ore passes now in use are 196 m. (642.88 ft.) long. Each of these openings delivers the ore to a cylindrical concrete ore bin, which has a capacity of 2200 short tons. Ore passes are driven at an angle of 60° to 62° and are lined throughout with steel H-beams placed in the form of cribbing. The ore passes are 5 ft. 0 in. by 6 ft. 0 in. in section and are widened to 20 ft. 0 in. at the top in order to conform to the requirements of the automatic dumping feature of the Granby-type cars.

Figs. 7 and 8 show details of steel lining in the ore passes. Control gates are placed at the junction of the ore passes with the ore bins, and also at intervals of 49 m. (160.7 ft.) in each raise. The ore contains from 3 to 4 per cent. moisture, and does not run well in the ore passes. If the opening is allowed to fill, the material packs tightly and it is difficult to get it started again; therefore the control gates are used, and after two trains amounting to about 200 tons have been dumped into an ore pass,

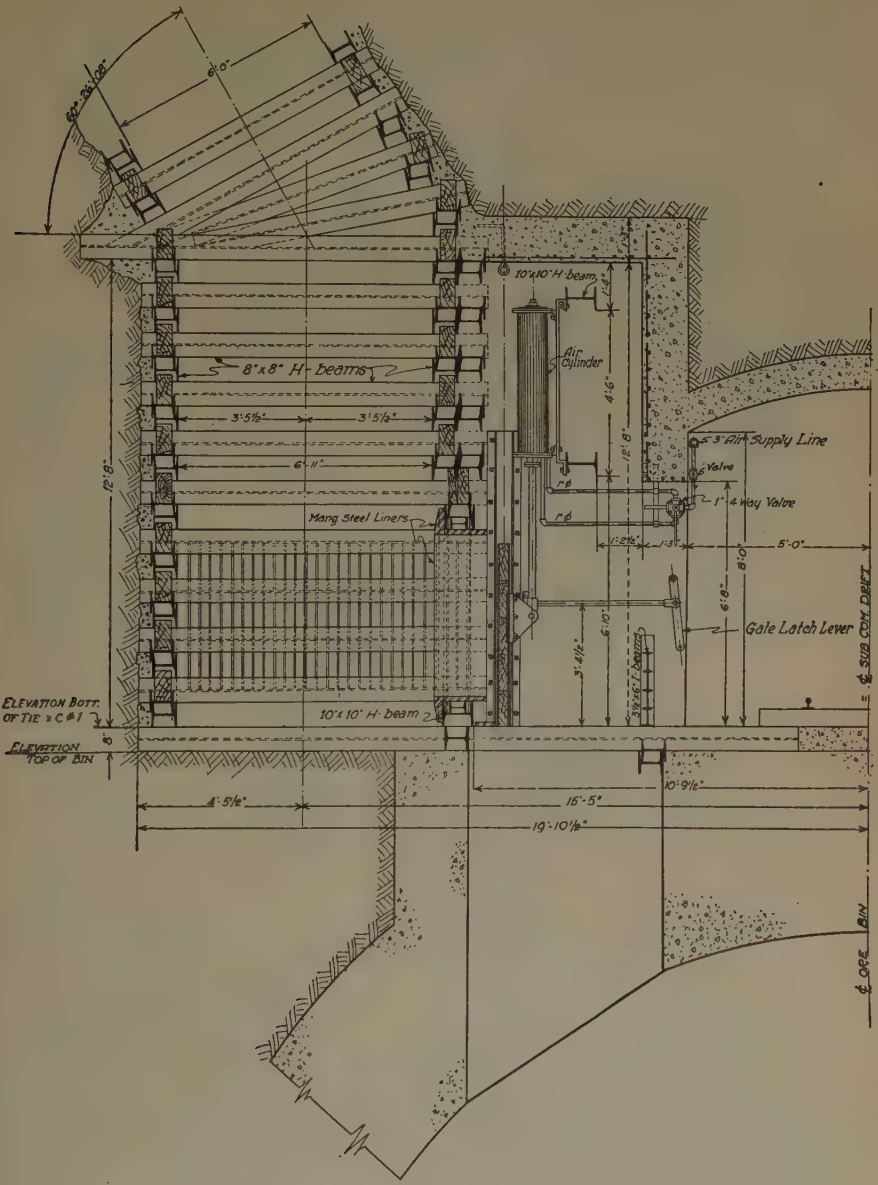

FIG. 7.—CONTROL GATE AT TOP OF ORE BIN.

the ore is dropped successively through the various control gates until it reaches the ore bin. The ground through which ore passes were driven is badly broken and fractured porphyry and any opening left without support would cave at once. Lining of the most enduring character must be used in these raises. For that reason the steel lining was developed and to date has proved satisfactory. All control gates on the ore passes are operated by 8 by 48-in. air cylinders having regular four-way valves. Communication drifts make connection with each ore pass at intervals of 24.5 m. (80.36 ft.) and each of these drifts connects with a supply shaft driven on the same incline as the ore passes. These drifts serve for inspection and repair of ore passes, and also provide means for reaching the passes at fixed intervals for purposes of breaking jams and keeping the ore moving.

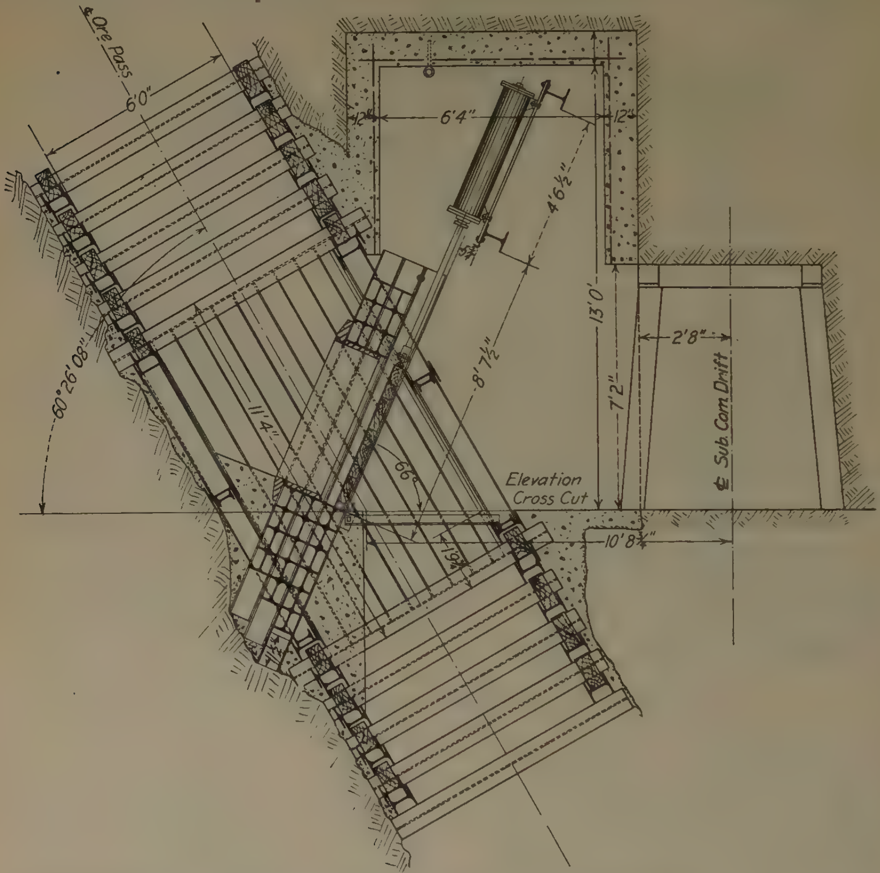

Fig. 8.—Inclined type of control gate used at subcommunication drifts.

Underground Storage Bins

The original design of the first ore-bin system called for the construction of six bins, two bins to be served by a single ore pass. On account of very heavy ground and difficult construction, two of these ore bins—Nos. 3 and 6 (Fig. 9)—were not built. Bins 1 and 2 are served by branches connected with one ore pass, while bins 4 and 5 each have individual ore passes. Ore bins of the original system are cylindrical in form, 8 m. (26.24 ft.) dia. and 25 m. (82 ft.) high.

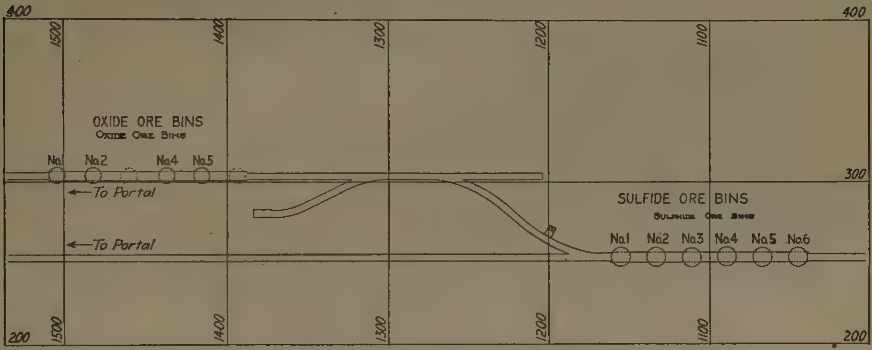

Fig. 9.—Part plan of main adit, showing layout of ore-bin systems.

A second system of ore bins made up of six units is under construction. On the new system, dimensions have been changed somewhat and bins are being constructed 10 m. (32.80 ft.) dia. and 16.5 m. (54.12 ft.) high. A dome-shaped structure is used at the top of the bins to support the ground above the bins, which is usually very heavy.

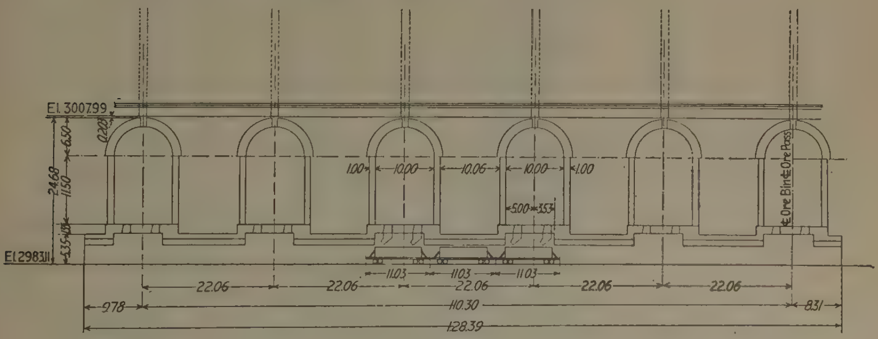

Fig. 10.—Longitudinal section on center line of sulfide-ore bins.

Ore bins are located directly above the haulage level and each bin is equipped with either two or three apron-type steel chute gates. These gates are controlled by air cylinders and counterweights. The counterweight serves for closing the gate and the gates open automatically when the counterweight is lifted by means of the air cylinder. Fig. 9 is a plan of the ore-bin system and connecting opening on main adit level, and

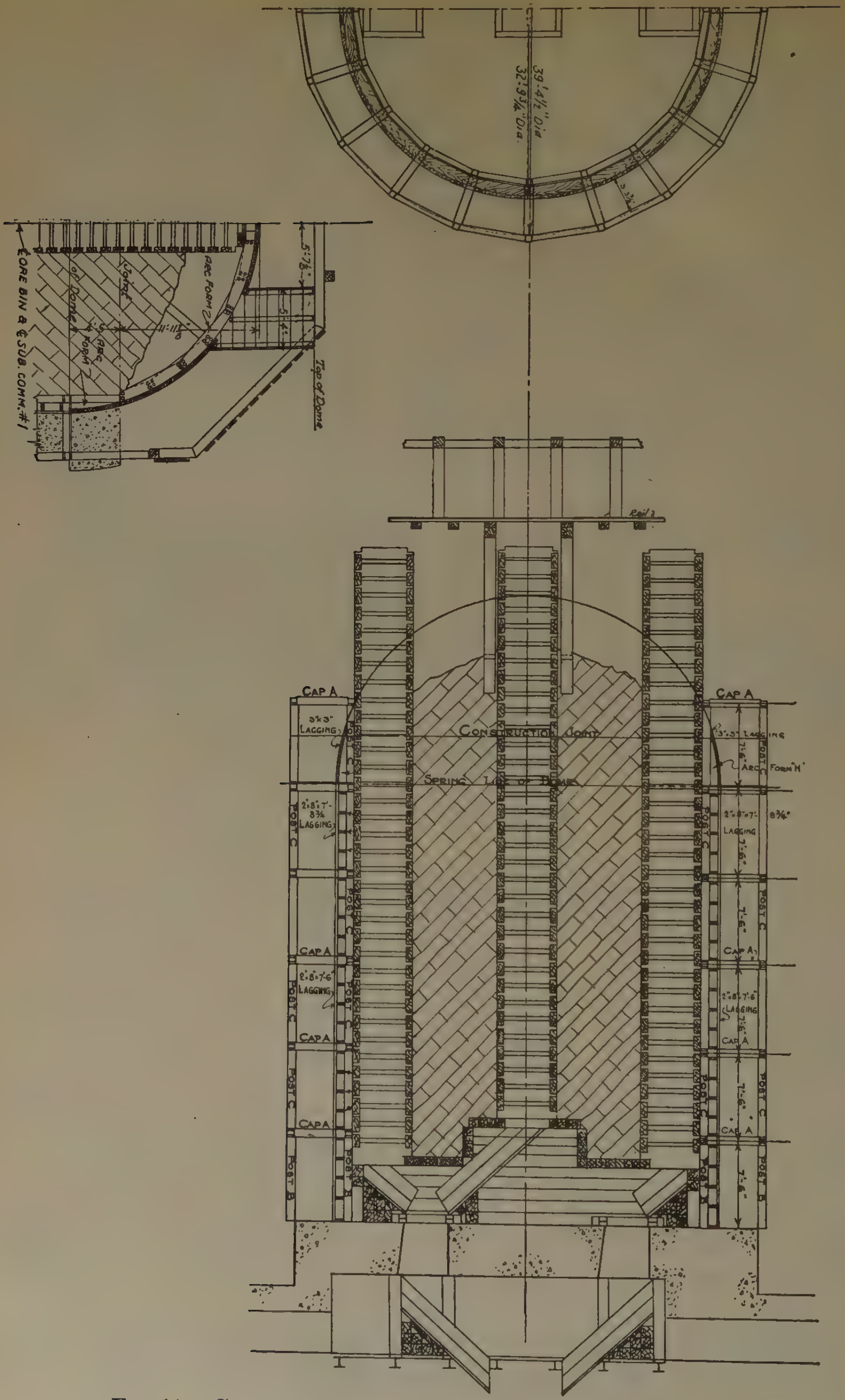

FIG. 11.—CONSTRUCTION TIMBERING, SULFIDE-ORE BINS.

Fig. 10 is a longitudinal section through the center of the sulfide ore-bin unit.

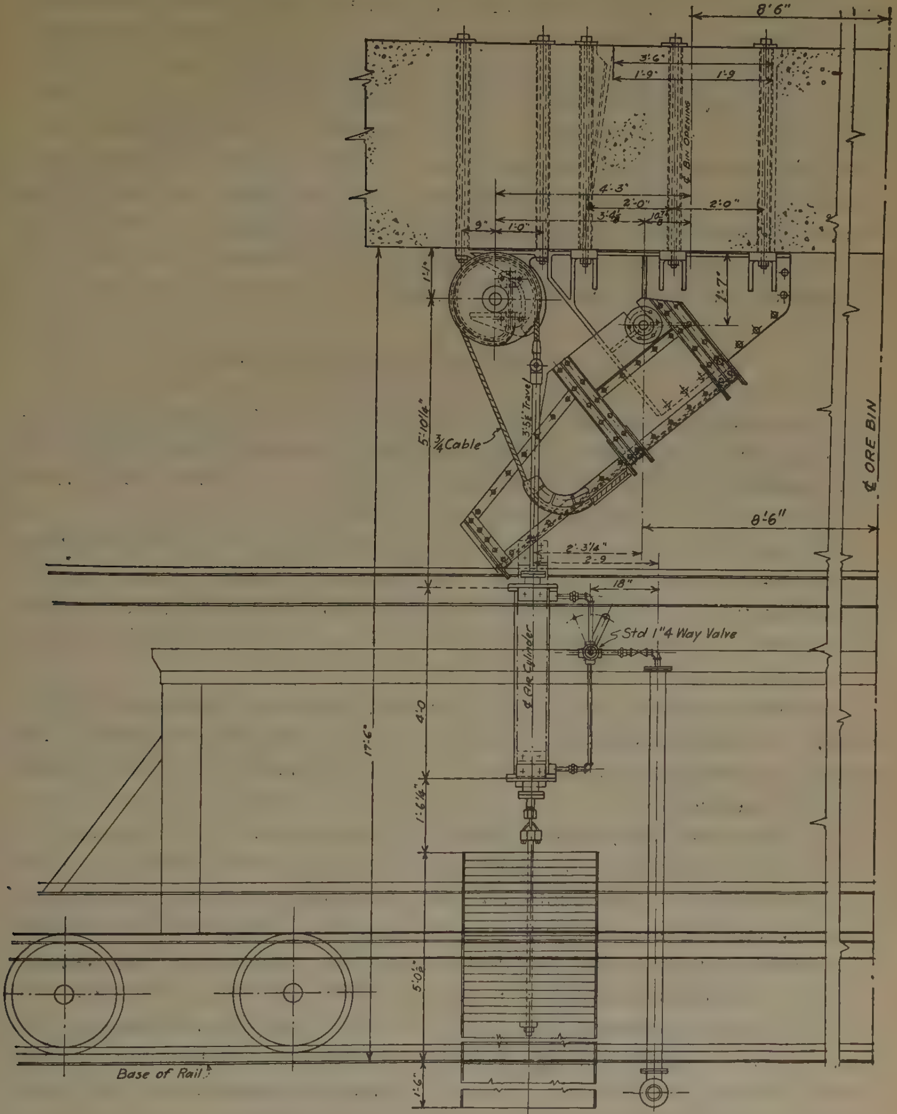

FIG. 12.—GENERAL ARRANGEMENT OF GATES AND OPERATING MECHANISM UNDER ORE BINS.

The system used for constructing these storage bins was as follows: The station under the bin was excavated and lined with concrete, then the concrete floor of the ore bin was placed. After the concrete floor had been allowed to set, a ring of square-set timbers was started on the floor, conforming to the periphery of the proposed bin. These timbers were carried up to the top of the bin, leaving the space inside of the timbers filled with broken material. This broken rock, inside the square-set timbers only, was then removed, leaving the center portion of opening filled (see Fig. 11), and the concrete walls were poured, using peripheral timbers for supporting forms. The concrete dome was put on, after which the material inside of the bin was excavated, starting at the top, and concrete forms were stripped as excavation proceeded downward. Alterations were made in dimensions of the ore bins when work was started on construction of the new unit, because the small diameter of the old bins permitted abrasion of the walls. Fig. 12 shows arrangement of apron gates and operating mechanism under the ore bins.

Mine Railway for Transportation of Ore

This railway, including the principal adit, was completed to the location of the first ore-bin unit in the year 1921. Concrete lining in the adit was finished in 1923 and no further work was done until 1925, when construction of the ore-bin system was started.

The adit and also other tunnels along this line have a section 10 by 11 ft.; that is, the tunnels are 10 ft. wide and 11 ft. from base of rail to the spring line. No timbering of any kind is used in any of these tunnels. Wherever it has been necessary to support the back, lining of concrete or rubble masonry has been constructed.

In the construction of tracks, 70-lb. rails were used and a rail of like weight is also employed for transmission and delivery of power. All trackage is laid out on a $3/10$ per cent. grade in favor of the load, curves being compensated.

The haulage equipment consists of 45-ton electric locomotives in tandem and Ingoldsby bottom-dumping cars of 40 tons capacity. Current is delivered to the locomotives at 500 volts, the current being supplied by two substations, one located near the coarse-crushing bins at Potrerillos and the other at the portal of the main adit (Las Vegas). Since the locomotives have been put in service they have been provided with trolley connections and a trolley wire is employed over all switches and at certain points on the main line where snow is inclined to drift deeply during the winter. Third-rail shoes on one side of the locomotives have been taken off, as a measure of safety, and the trolley has been substituted.

Trains are loaded under the ore bins without spotting the cars and without bringing the trains to a stop. As the empty cars at the forward end of the train pass under the first or most southerly ore bin, a certain amount of ore is drawn into the moving cars, but no attempt is made to load them completely. As the cars pass under the other bins, additional ore is dropped into each car and finally the load is finished and trimmed as cars pass under the final bin. Ore can be transferred from storage bins to coarse-crushing plant at a rate of 1600 tons per hour, using two trains.

UNLOADING AND DELIVERY OF MATERIAL

At the portal of the main adit is located a camp known as Las Vegas, where workmen have been housed during the construction period. This townsite will be maintained in the future for the benefit of trainmen engaged in the transportation of ore, trackmen, material handlers, etc. All materials and supplies to be used in connection with mining operations are transported by rail to this point (Las Vegas). From here timber, explosives, steel, etc. are hauled by trucks about 4 km. (2.48 miles) over a dirt road extending to the main mine camp.

Because work is going forward at many different points and on different levels and because most of these levels connect with the surface, we have found it cheaper to transport material on surface directly to portals of the various adits than to make use of underground transportation by way of the main adit and supply shaft. From 3000 to 5000 tons of material is handled during the course of each month in this manner and trucking costs are extremely low.

UNDERGROUND EQUIPMENT

In connection with underground work on the various levels, 19 trolley-type 10-ton locomotives are in use. All take current at 250 volts. In addition, four storage-battery locomotives of 6½ ton weight are used in the handling of development work. Two hundred Granby-type ore cars of 5 tons capacity are used on the intermediate haulage level, also 150 rocker dump cars which are used for the transportion of waste and ore that must be stored on surface.

Air is supplied to all workings from a central compressor plant containing two Nordberg compressors each of which has a capacity of 5200 cu. ft. of free air per minute, sea-level rating. There is also one Ingersoll-Rand compressor with a capacity of 2600 cu. ft. free air, sea-level rating. The necessary motor-generator sets for furnishing direct current to the locomotives are in the same building.

Sharpening of drill steel and tools is accomplished at a number of points. The principal steel-sharpening shop is underground on the intermediate haulage level, but three additional steel-sharpening shops are used because the work is so widely scattered and we have found it inadvisable to transport steel to a central sharpening plant. For sharpening of drill steel three No. 50 and one No. 5 Ingersoll-Rand sharpening machines are employed.

A complete machine and repair shop is located at the portal of the intermediate adit. The equipment consists of the usual machine tools and all repair work for the mine is done at this point except the making of castings. For this class of work we depend on the foundry at the reduction plant.

CONCLUSION

It has been the aim of this paper to cover in a general way only the important features connected with development and underground construction. Production of sulfide ore has been under way for only a little more than one year and the first block to be developed has not yet been fully mined. Mining of oxide ore is to be started during the current month (April, 1928). Information on mining costs has not been given here for the reason that operations are virtually in the preliminary stage and costs are not as yet stabilized.

DISCUSSION

G. S. RICE, Washington, D. C.—Is Fig. 3 a characteristic vertical cross-section of the orebody, and if it is, has the work developed enough to know whether the quartz diorite porphyry hanging wall shows any symptoms of breaking off when the underlying ore is extracted?

C. E. ARNOLD, New York, N. Y.—I gather that you refer to the contact between the porphyry and the sedimentary?

E. S. RICE.—Yes, I refer to breaking above that contact.

C. E. ARNOLD.—While there is a gouge along that wall and it is quite soft, and has been the course for descending solutions, when breaking starts it travels practically vertically to the surface regardless of that contact shown.

G. S. RICE.—You spoke about the pressure requiring the use of concrete lining of the grizzly drifts. Was that concrete reenforced?

C. E. ARNOLD.—Yes, but I would say that not as much of the concrete work has been used as of the rubble masonry, and I think they prefer the rubble masonry to the concrete.

G. S. RICE.—If this (Fig. 3) is a typical section, does that pressure come wholly from the broken ore or does the whole ground above the undercutting levels produce more or less pressure? I notice that the hanging wall is about 200 m. high to the surface above the ore passes.

C. E. ARNOLD.—I do not know, but I would say that it is a combination of both.

G. S. RICE.—How badly are the H-beams in the ore passes abraded by the ore?

[C. E. ARNOLD.—At first, when they tried to run an ore pass as one continuous chute, the H-beams wore very fast. Wear was immediately noticeable, so that early warning was given and steps could be taken to prevent the situation from becoming serious. By splitting the ore passes into 150-ft. sections, approximately, and treating each section as an ore-pass unit, the wear is greatly reduced.

G. S. RICE.—Are the ends and the sides tied together by angle irons?

C. E. Arnold.—No, they are not tied. They are put in as cribbing, but, of course, when the footwall and hanging-wall members of the cribbing are put in, a spreader is put in the trough of each of the two side H-beams, which keeps the hanging-wall and the footwall cribbing members apart; spreaders are placed in the troughs of the footwall and hanging-wall cribbing members to keep the side members apart.

G. S. Rice.—Have you any figures indicating the tonnage per man underground?

C. E. Arnold.—For direct labor underground they have been getting during the last few months from 28 to 33 tons per man.

H. G. Moulton.—You ought to get better than Miami because you do not have to control as closely, which would bear out these figures.

C. E. Arnold.—In the particular place where we are getting that tonnage we have to control probably just as closely as Miami, but where the drawing is done without any overburden we expect to better those figures.

G. S. Rice.—That, I suppose, does not include the men on development and the men putting in the branch raises?

C. E. Arnold.—Yes, including the developing of that particular block.

G. S. Rice.—Including putting in the timber and concrete?

C. E. Arnold.—Yes, that is direct labor.

H. G. Moulton.—Do you put the square sets in on the top by contract or is it done by day labor?

C. E. Arnold.—Practically all of the work that can be contracted is contracted

H. G. Moulton.—And you do that at so much per set?

C. E. Arnold.—Yes, at so much per set.

G. S. Rice.—What proportion of the men are English speaking? Are they miners brought in from America, or other experienced miners?

C. E. Arnold.—I do not know. The proportion is very small. The only men imported into the country are the superintendents, assistant superintendents, engineers, geologists, mine foremen, and some shift bosses.

I. A. Ettlinger, New York, N.Y.—Is Mr. Arnold prepared to give costs?

C. E. Arnold.—I regret that I have no authority to give costs.

H. G. Moulton.—What powder are you using down there, Mr. Arnold, 60 or 40?

C. E. Arnold.—I think 35 per cent.; that is, in the caving operations.

H. G. Moulton.—You are not doing any liquid oxygen blasting, are you?

C. E. Arnold.—No.

H. G. Moulton.—Do any of the other companies in South America use the Granby car?

C. E. Arnold.—So far as I know, no.

H. G. Moulton.—It is a car that is entitled to be used more than it is. It is the principal contribution of Granby to mining technique, and I have been rather surprised that its use has not been more extensive. Have you had occasion to use it in sticky material, or wet material?

C. E. Arnold.—Yes, we use it in the stickiest ore we have, though it is not really "sticky" ore. The cars clear a good deal better than hopper-bottom cars.

H. G. MOULTON.—You have no difficulty in keeping them very tight against spilling of fine material?

C. E. ARNOLD.—No.

H. G. MOULTON.— Do you not think that the coal men ought to use them more than they do?

C. E. ARNOLD.—I do not know. These cars are well adapted to loading from chutes. It would be out of the question to shovel into them, because they are too high.

H. G. MOULTON.—Yes, for shoveling, but for transfer they would work well.

MEMBER.—What is the height of the car?

C. E. ARNOLD.—About 6 ft. from the base of the rail.

H. G. MOULTON.—Of course, height is not a fundamental part of the design. It can be varied and altered. You can apply the same design to almost any relative proportions in the size of the car.
Are they running both oxide and sulfide, Mr. Arnold?

C. E. ARNOLD.—Yes.

H. G. MOULTON.—What is the general method of oxide treatment?

C. E. ARNOLD.—It is straight sulfuric acid leaching. The enriched solutions are purified, and delivered to the electrodeposition department, where they make fine copper cathodes which are melted and cast into wire bars.

H. G. MOULTON.—You do not attempt to recover any of the sulfide in the mixed ore by leaching?

C. E. ARNOLD.—No, we have no mixed ores. There is a very sharp transition from the oxide ore to the sulfide.

G. S. RICE.—Are you making plans to study the question of subsidence with reference to the extraction of deeper sulfides? Are you carrying on surveys for that purpose?

C. E. ARNOLD.—They are watching that very closely, because there is one portion of the sulfide ore that has a depth of about 275 m., and there is a considerable amount of barren sedimentary high on the flank. It is a question of how much of that overlying sedimentary material we can afford to move, and one of the points that must be considered is what the ultimate caving slope will be.

G. S. RICE.—According to Fig. 3, I suppose that the critical side would be on the porphyry.

C. E. ARNOLD.—It is not to the section shown in Fig. 3 that I have reference; the section I refer to is much farther north in the mine. The sedimentary has the same dip as shown in Fig. 3 but lies on the opposite side of the porphyry. The hill runs up steeply to the left-hand side, and before we cave the sulfide at the bottom we want to know where the plane of breaking is going to lie from the edge of the caving workings to the surface.

Subsidence from Block Caving at Miami Mine, Arizona*

By F. W. Maclennan,† Miami, Ariz.

(New York Meeting, February, 1929)

Papers by D. B. Scott, E. G. Deane, and J. H. Hensley, Jr.[1] describe the succession of mining methods used in the Miami mine—square-set system, shrinkage stoping, top-slicing method, and the undercut caving system which was then being used, and which embraced the use of finger raises. The last-named system worked well but when the rich ore was exhausted it became necessary to reduce costs so as to make available a low-grade ore containing an average of less than 0.9 per cent. total copper with less than 0.8 per cent. in sulfide form and with no precious-metal values.

There was then developed for this low-grade ore the system of separate large block-caving stopes, in lateral dimensions 150 by 300 ft., or 150 by 150 ft., with intervals between stopes of the same size so that subsequently these pillars may be similarly mined. The results have been most favorable. Under the previous system the production of ore per man-shift for all underground labor was 8 tons. The present production per man-shift underground is 28 tons. In starting this work the problems were:

1. How the ground would cave above the stopes.
2. Whether in the first stopes caving would encroach on the pillars as laid out.
3. Whether in subsequent mining of the pillars waste filling would be drawn from the previously caved adjacent stopes.

Fig. 1 is a small key plan for showing the location of cross-sections as well as of ore mined and the limit of the escarpment and limit of cracking on the surface caused by the caving of the capping overlying the ore withdrawn. Very little caving is found outside of the boundary of the stopes up to the top of the ore. Special care has been taken to prevent caving beyond the stope boundaries by driving boundary caving drifts at suitable vertical intervals, to cause the stopes to cave to this boundary and not beyond it.

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* The following papers on subsidence will appear in a volume of *Transactions* on Coal and Coal Products to be issued in 1930:

W. Herd: Bumps in No. 2 Mine, Springhill, Nova Scotia.

H. W. Montz: Subsidence from Anthracite Mining. With an Introduction on Surface Support, by R. V. Norris.

H. Louis: The Royal Commission on Mining Subsidence.

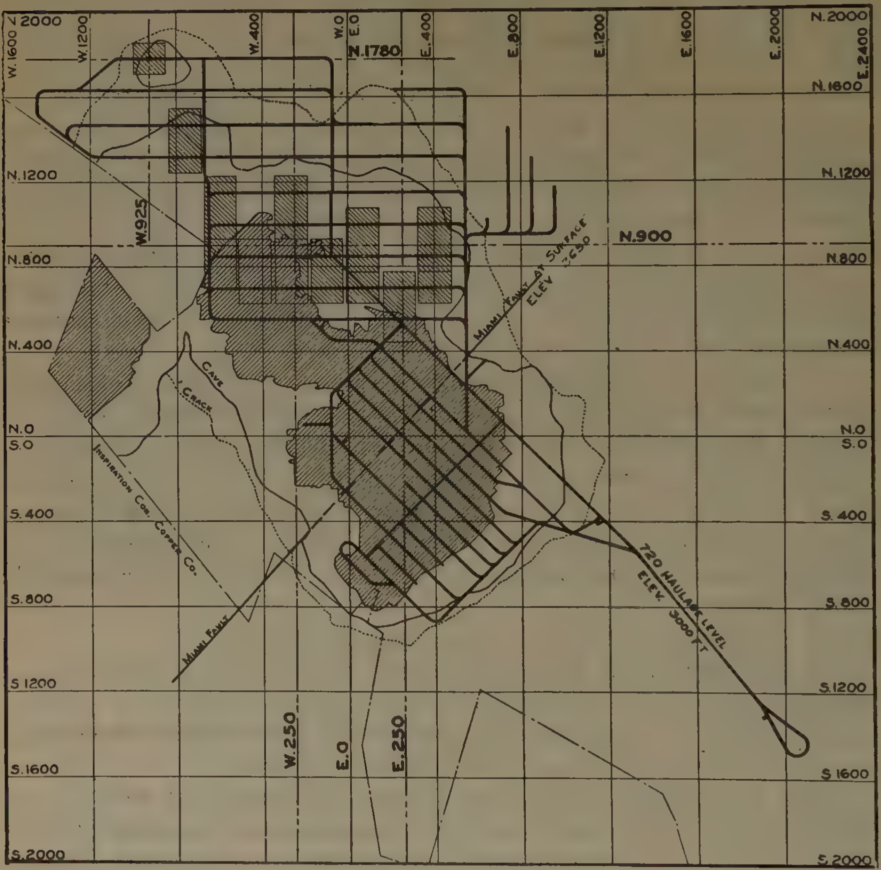
† General Manager, Miami Copper Co.

[1] D. B. Scott: Stoping Methods of Miami Copper Co. *Trans.*, A. I. M. E. (1916) **55**, 137.

E. G. Deane: Block Method of Top Slicing of the Miami Copper Co. *Idem* (1916) **55**, 240.

J. H. Hensley, Jr.: Mining Methods of the Miami Copper Co. *Idem* (1925) **72**, 78.

From the results so far it is believed that there is little caving or ground movement beyond the boundaries in the orebody. There has been only one instance where any movement has been found in the ore adjacent to stoping operations, and that is shown on the East 250 section by crack E, which records the movement of a few inches of the drifts on the various levels.



Scale: 1 in. = 540 ft.

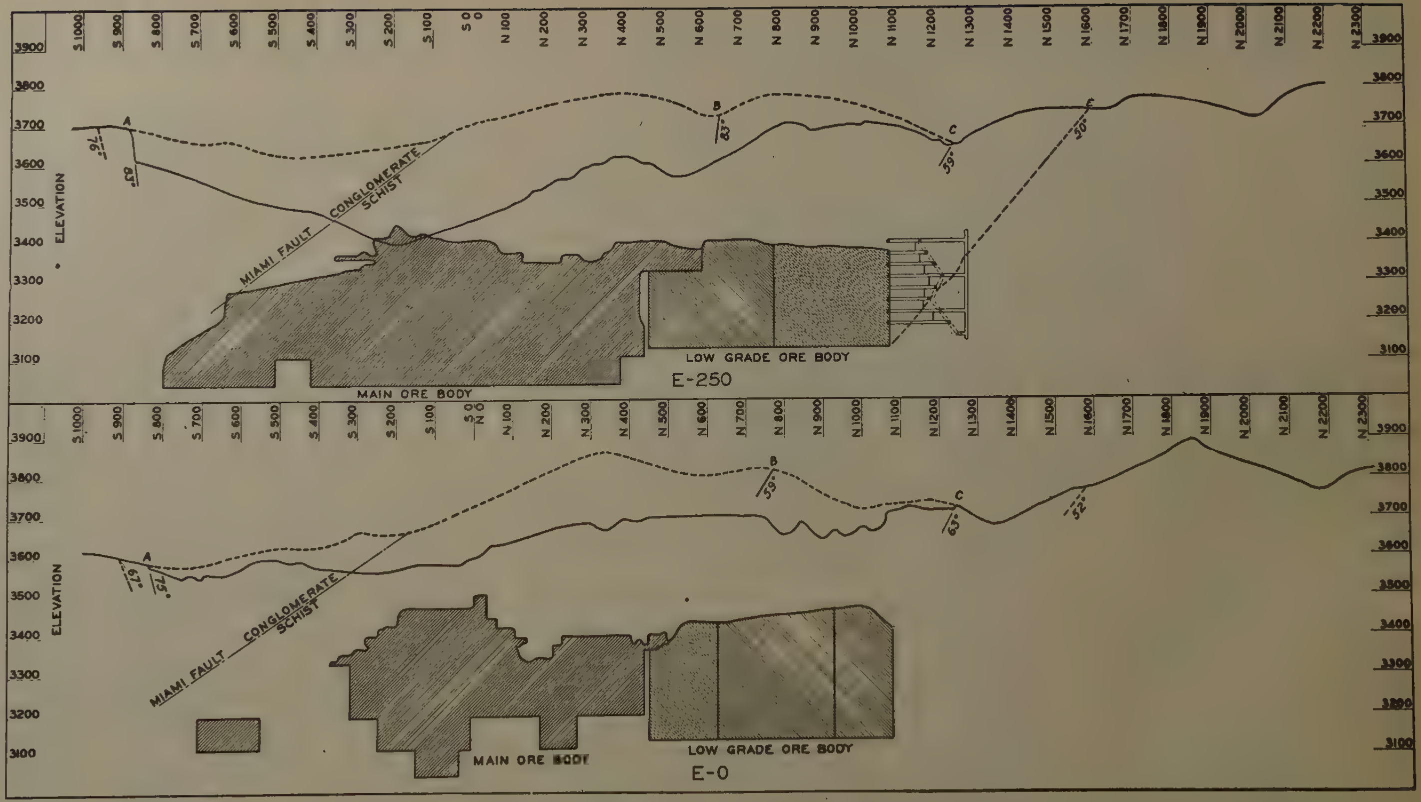
Fig. 1.—Plan of 720-ft. haulage level, showing caved areas and resulting surface subsidence.

So it may be that in our case the angle of draw should begin with the top of the orebody instead of the bottom. This is quite an important point in mining the ore, as alternate stopes are utilized and the original stopes are depended upon not to encroach on the pillars and the pillars not to move laterally towards the original stopes.

Figs. 8–12, inclusive, are photographs taken from the same place showing the settling of the surface from Jan. 1, 1926, to Nov. 1, 1928. The difference in the subsidence shown over the area in the first and last photographs was caused by the removal of 11,400,000 tons of ore. The subsidence caused by the mining of an individual stope, the horizontal dimensions of which were 150 ft. square, will be noted on the plan (Fig. 1) and also on sections N. 1780 and W. 925 (Figs. 2 and 3). The time element in the subsidence of the surface over this small area is of interest. The area was undercut at the bottom of the ore at an elevation 650 ft. vertically below the surface. This overburden consisted of a vertical height of 220 ft. of highly fractured ore, which has a tendency to cave much more easily than the average ore at Miami, so much more so that it was necessary to use a method of undercutting that did not involve opening such large areas as usual in the regular system of undercutting. The 220 ft. of ore was overlain by 430 ft. of capping, which I presume would also be more easily subject to caving than the average of orebody at this mine. This area, known as stope 11, was completely undercut and full-scale, ore-drawing operations commenced on June 14, 1928. On Sept. 25, 1928 (104 days later) this stope caved through the surface, and up to this time, 115,000 tons had been drawn. This tonnage is equivalent to a vertical depth of ore of 82 ft. of the entire area of the stope. In other words, the entire vertical column of ore and waste of 650 ft. subsided through undercutting of the bottom and removal of 12.6 per cent., or approximately one-eighth of its volume. Up to Dec. 1, a vertical depth of 131 ft. of ore was drawn, causing a maximum surface subsidence of 60 feet.

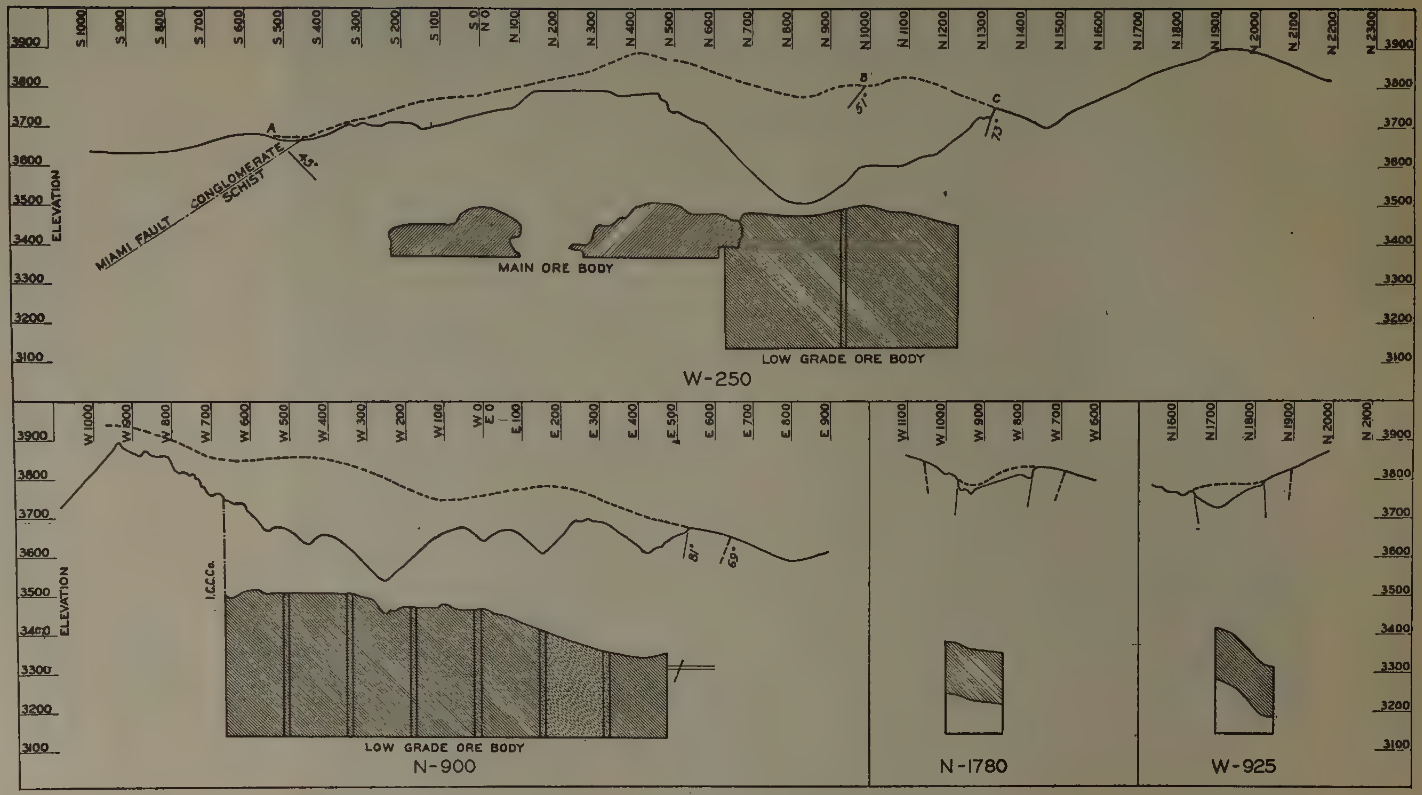
It is also interesting to note that this stope has caved almost vertically over the area undercut, veering off to the southwest about one-half the width of the stope, and that the area of subsidence on the surface is not very much greater than the area of the stope. The surface vertically over the space between this stope and the adjoining stope, which is almost completely drawn, is absolutely intact, with no subsidence showing in spite of the fact that the two stopes are only 150 ft. apart and are silled off 650 ft. below the surface.

The extreme dimension of the orebody north and south is 2700 ft. and east and west 2100 ft. The maximum subsidence is 270 ft. shown on the W. 250 section at N. 850, this subsidence resulting from the removal of a vertical depth of 340 ft. of ore. This represents a subsidence of 79.4 per cent. of the depth of ore removed, which seems rather high, but there are so many contributing factors, such as the previous mining of ore to one side of this section, causing part of the overlying surface to slide



Scale: 1 in. = 118 ft.

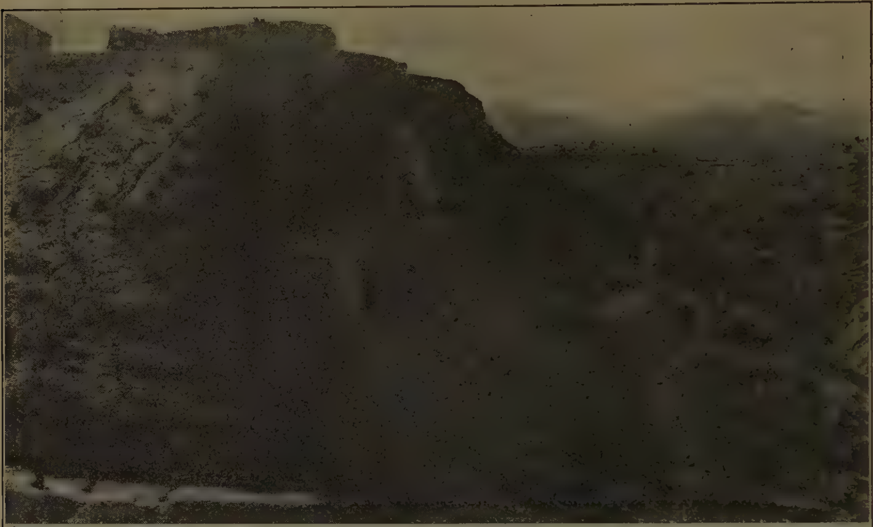
FIG. 2.—SECTIONS THROUGH OREBODY SHOWING STOPED AREAS AND RESULTING SURFACE SUBSIDENCE.



Scale: 1 in. = 118 ft.

FIG. 3.—SECTIONS THROUGH OREBODY SHOWING STOPED AREAS AND RESULTING SURFACE SUBSIDENCE.

off horizontally, thus reducing the amount to be subsided, that it is not safe to draw conclusions from these data. In an estimate made some years ago, by figuring the volume of ore mined and the volume of subsidence, the average appeared to be that the subsidence volume represented about two-thirds of the volume of ore removed.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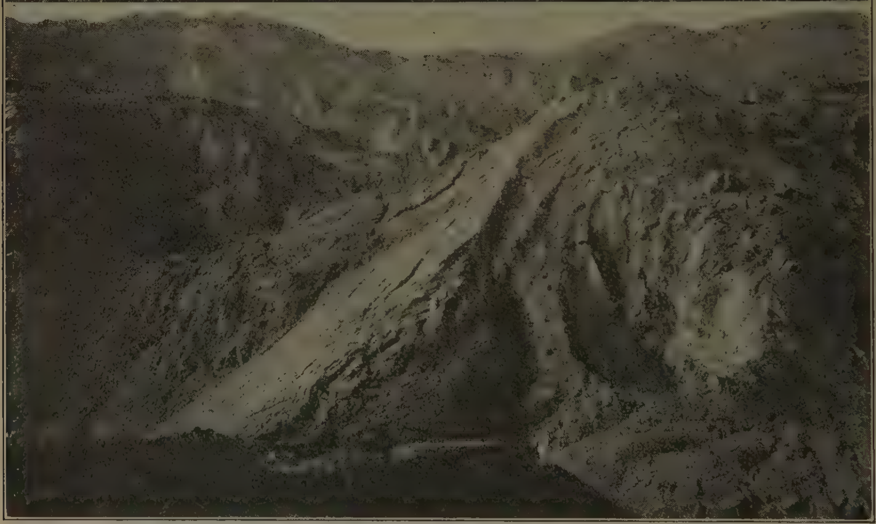
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Fig. 4.—Escarpment on conglomerate showing the nearly vertical break.
Fig. 5.—General appearance of cave over low-grade orebody.

Description of Illustrations and Details of Ground Movement*

The accompanying photographs and drawings illustrate the nature and extent of subsidence caused by mining operations at the Miami

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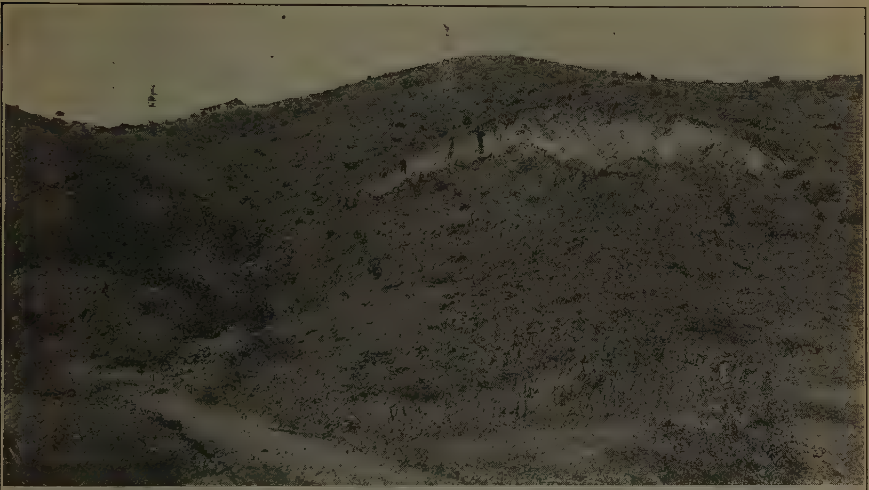
* Prepared by E. V. Graybeal, Mining Engineer, Miami Copper Co.

Copper Co. The caved area over the orebodies is entirely in schist except along the Miami fault where the hanging wall is in conglomerate (Fig. 4). The general appearance of the cave over the low-grade orebody is shown by Figs. 5 and 7. Figs. 8 to 12, inclusive, are photographs taken from the same point covering a period of about 3 years, and show the drop in successive stages. Fig. 1 is a plan of the 720-ft. haulage level; the hatched area indicates the mined portions of the orebodies; surrounding this area the solid line marked "cave" represents the escarpment on the surface resulting from these operations and the dotted line outside the cave designates the location of any further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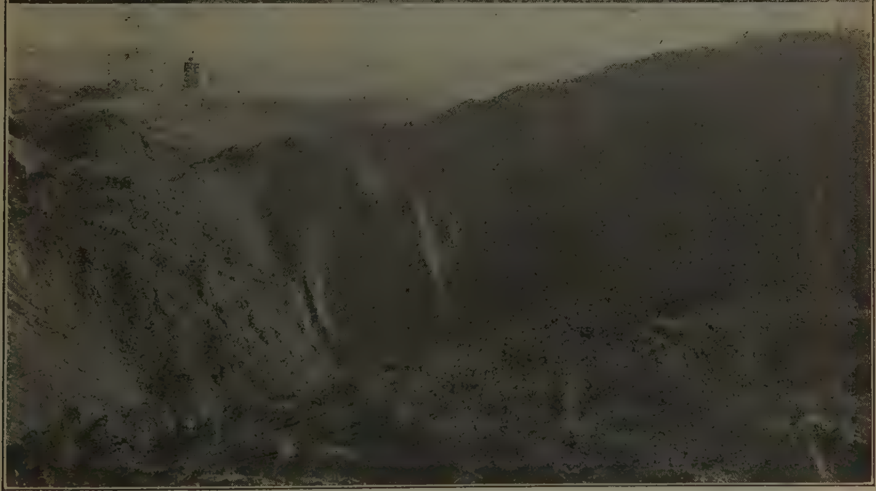
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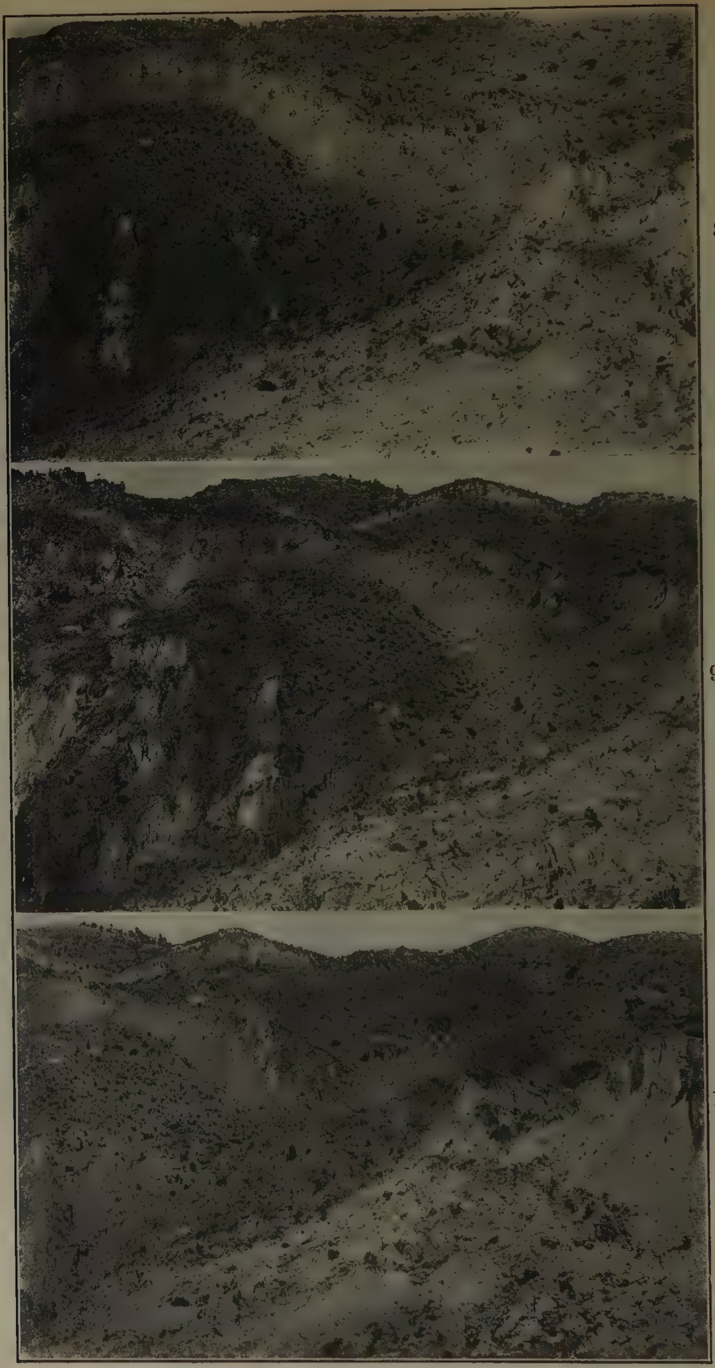
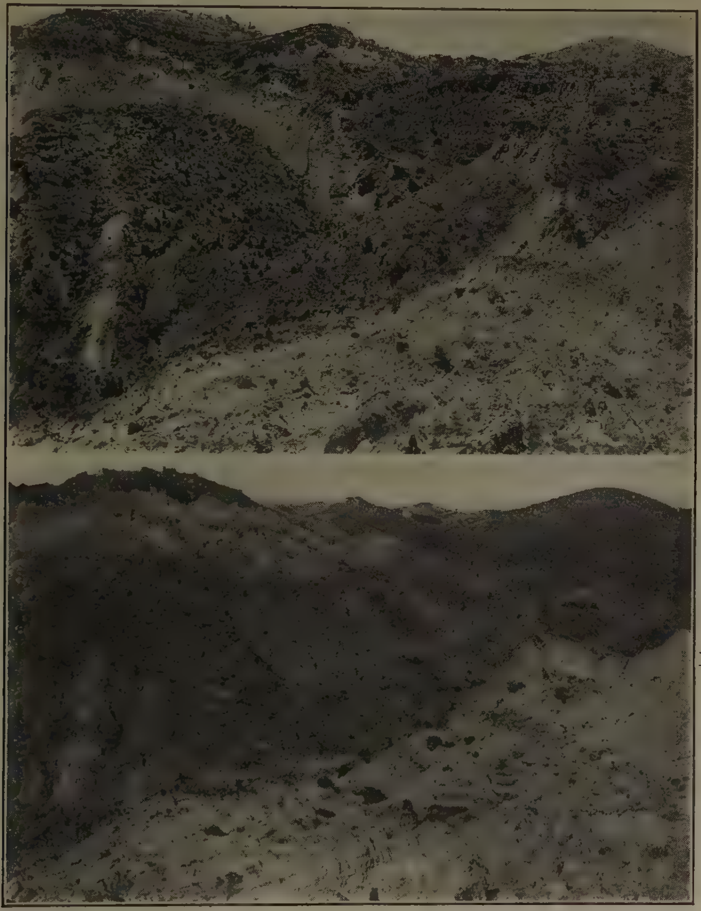
FIG. 6.—SUBSIDENCE OVER STOPE SHOWN IN N. 1780 AND W. 925 SECTIONS.

FIG. 7.—LOOKING SOUTH ACROSS CAVE OVER LOW-GRADE OREBODY. THE CONCENTRATOR AND TAILING PILE ARE IN LEFT BACKGROUND.



Figs. 8–10.—Successive stages of subsidence at Miami mine, Jan. 1 to Apr. 29, 1926.

movement, usually small disconnected cracks. The sections (Figs. 2 and 3) show the position of the orebodies with respect to the surface and the subsidence resulting from their extraction. The original surface is shown dotted over the present surface. The areas hatched with solid lines represent ore mined and the areas hatched with dotted lines are adjacent to exhausted stopes; the subsidence over these areas is therefore caused by drawing on either side of the section.



Figs. 11 and 12.—Subsidence at place shown in Figs. 8–10, June 15, 1926 to Nov. 1, 1928.

The capping varies in thickness from 245 to 500 ft.; the ore has an average thickness of 315 ft. and is 81 acres in extent. Both ore and capping are Pre-Cambrian schist, the character of which varies in different parts of the mine, as there has been much alteration due to extensive fracturing and silicification as well as kaolinization along numerous small faults and slips. There is very little ground water and this is drained off or evaporated during development preparatory to stoping. The susceptibility to caving depends largely on these factors as well as upon the area undercut.

The main orebody was mined by a block caving system, the details of which have been previously published.[2] Successive lifts of about 75 ft. in height were undercut and drawn beginning at one side and retreating across the orebody. The resulting subsidence is shown on the sections (Figs. 2 and 3) between points *A* and *B*. The break followed the drawing closely and very little piping through to the surface occurred.

The low-grade orebody is mined at present by a system in which the orebody is divided into stopes either 150 by 300 ft. or 150 by 150 ft. in area. The entire head of ore is taken with one undercutting operation and lateral caving is prevented by boundary drifts driven around the perimeter of the stopes at 30 or 45-ft. vertical intervals, depending on the ground. Any required number of pillar stopes may be drawn after sufficient time has elapsed to allow the caved capping to consolidate in the exhausted stopes. On the sections (Figs. 2 and 3) the caved area joins the main orebody cave on the south and extends north to point *C*. There is probably very little caving outside the boundaries of the stopes themselves; numerous places for observing this action are provided by the development drifts of adjacent stopes. However, slight movement outside the stope boundaries may develop, as is shown by the crack E on the E. 250 section (Fig. 2). The ore has been extracted to N. 1075 on either side of this section and as a result of these operations the crack, with several inches of movement, occurred in the drifts to the north. On the surface the crack is about 1 ft. wide. On the W. 250 section there is no apparent movement underground outside the stope boundaries, and on the surface no cracks appeared in advance of the escarpment at *C*. The plan shows a crack to the north of this escarpment, but this crack appeared at a much later date and was caused by stoping west of this section. The N. 1780 and W. 925 sections are at right angles through the center of an isolated stope from which about 130 ft. of ore has been drawn. Fig. 6 is a photograph of the cave over this stope. The subsidence has been unusually uniform, and with no piping to the surface, as is usual with this method of mining.

DISCUSSION

G. S. Rice.—Mr. Maclennan has given us one of the most interesting and valuable studies that has been made public on ground movement and subsidence from metal mining, because it deals with a new layout of block caving stopes and because it gives information on subsidence based on accurate surveys.

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[2] J. H. Hensley, Jr.: *Op. cit.*

In going over the rugged surface above the mine a year and a half ago, I observed that monuments had been placed to show the position of each corner of each of the separate stopes in low-grade ore. I learned that surveys were being made from time to time giving information on the time of the movement as well as the amount of movement. A number of points of exceptional interest deserve comment.

Referring to the "main" orebody mined previously, as shown in the plan of Fig. 1, and in the cross-section of Fig. 2, it is interesting to note that although a strongly marked fault ("Miami" fault), indicated as following the contact between the conglomerate and schist, intersects diagonally the overburden above the south part of the mine, the angles of surface "break" or "draw," varying from 67° to 76° from the bottom of the workings, cut across the plane of the fault, apparently without the fault affecting the angle of draw either in the direction at right angle to the fault or parallel with it.

On the other hand, in the northern part of the mine, the angle of draw extends from the bottom of the low-grade ore stopes adjacent to the main orebody, as shown in sections E. 250 and E. 0 (Fig. 2) at angles of only 50° to 56° respectively, and these appear to be in planes approximately parallel with the Miami fault.

Are these flatter dips parallel with the bedding planes of the schist and is this the reason that the angles of draw are so low? Such effect is not indicated by section N. 900 (Fig. 3) where the angle of draw appears to be at a steeper angle, 69°. However, the east and west direction of this vertical section apparently does not register the maximum draw, which appears to lie to the northwest of the low-grade orebody where the full line that marks the escarpment indicates a much flatter draw than 69°.

In section W. 250 (Fig. 3), which runs in a north and south direction, the angle of draw, 45°, appears to be the flattest recorded. It extends southward from the bottom of the main orebody, in a direction apparently at right angle to the bedding planes of the schist.

Most interesting sections are those of the two isolated stopes in the low-grade ore as shown in N. 1780 and W. 925, Fig. 3. The angles of draw are steep and apparently at this earlier stage of subsidence the bedding of the schist has little influence upon the draw.

In the stopes underground I was greatly impressed to see how little ground movement there was, as Mr. Maclennan mentions, of each stope beyond its respective boundary, as shown in the mine workings. This was probably in large part due to the care with which each stope was cut around, so that in effect the descending ore acts as a piston column, the broken ore supporting the walls of ore at the sides.

That the system of mining has been most successful is well known to the metal-mining industry. It is indicated by the large yield per man-shift, and it has made highly profitable the mining of an ore so low in grade that formerly it would have been considered worthless rock.

Mr. Maclennan may now be in a position to tell us about the probability of recovering most of the ore in the intermediate blocks without weakening by waste drawn in from the adjacent filled stopes. In any case the development of this method of block caving has been a splendid success.

F. W. Maclennan.—I believe the dips at the north end of the orebody are flatter largely because they are parallel with the bedding planes of the schist, while the steeper dips at the south end of the orebody cut across the bedding planes of the schist. The schistosity dips approximately south at an angle varying from 25° to 75° from the horizontal.

Regarding the flattest angle of draw, 45°, undoubtedly this is due to the fact that a considerable amount of ore was drawn a little to the east of this section and considerably farther to the south than the point from which the 45° dip is started south.

Referring to the small, isolated stope, the boundaries of the first caves to the surface are usually nearly vertically over the boundaries of the ore removed underground, and as the ore is drawn down, the sides of the cave slide in and take an ultimate angle similar to the ones shown on the sections.

Referring to the last paragraph of your discussion, several of the pillar stopes have already been worked out with results that compare very favorably with the original stopes both as to costs and extraction. The first pillar stopes mined out are not so good, but the later ones compare favorably with the average of our original stopes and are better than some of them. With the care we use in cutting the original stope from the pillar and the close control we apply in mining both the original stopes and the pillar stopes, I am confident that satisfactory results in the pillar stopes are assured.

[During the session at which this paper was presented, C. W. Wright mentioned a paper on subsidence in metal mining—W. R. Crane: Subsidence and Ground Movement in the Copper and Iron Mines of the Upper Peninsula, Michigan. U. S. Bur. Mines *Bull.* 295.]

Handling of Fine Ores and Concentrates in Salt Lake Valley Lead Smelters

By L. D. Anderson,* Salt Lake City, Utah

(New York Meeting, February, 1929)

When, after years of troublous experiences in roasting sulfide ores with heavy dust and fume losses resulting from the equipment and methods first available, there appeared on the scene of metallurgy that revolutionary process known as sintering, it was felt that at last there had been found the ideal method for preparing sulfide ores for blast-furnace reduction. Not only were the sulfides roasted but also the fine oxidized ores were bound together with them in strong porous cellular masses which were in the best possible condition for reduction. With the experiences gained in the manipulation of the Huntington-Heberlein, Carmichael-Bradford, and Dwight & Lloyd processes there developed a refinement of detail which brought the whole plan of roasting by sintering to a high plane of perfection. But that metallurgy is never stationary is revealed by the fact that changes in milling practice, particularly the development of flotation, have introduced a new set of problems in sintering itself. Inasmuch as it is felt that these problems have not yet been solved in an entirely satisfactory manner this paper is written to record the present practice of the Salt Lake Valley smelters with the hope that there will be brought forth such discussion as may perhaps lead to still better practice.

By far the greater proportion of the sulfides smelted today are flotation concentrates. Usually they are high grade, sometimes as high as 70 per cent. and more in lead. They are extremely fine, many mills grinding the ore so that from 60 to 80 per cent. will pass through screens of 200 mesh. The physical condition in which many of these concentrates reach the smelters is disconcerting to the metallurgists. Some are of about the consistency of putty. Others are of a sticky, slimy nature. Others again, dried to the limit, are veritable powders. Remembering that the down-draft sintering charge for the best results must be porous enough to let air be drawn through it reasonably well, must be thoroughly mixed, and must have some coarse material in it to segregate out and form a layer on the grates, it will be understood that the advent of this extremely fine material has made necessary still further elaboration in the equipment and processes of preparing sulfides for the blast furnaces, all of which is expensive. The high metal contents of the concentrates have brought about another difficulty caused by the incipient fusion of the lead which caused the material to frit together, close off the passage of air and end the roasting operation before the necessary amount of sulfur had been removed. In addition the resultant sinter was weak and friable so that the blast furnaces which had been running freely with the old hard sinter from crude sulfides were choked. The manner in which the difficulties outlined have been coped with thus far is a record of real achievement.

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* Chief Engineer, United States Smelting, Refining & Mining Co.

Double Roast

Probably the biggest step in the solution of the problem came in the development of the "double roast," wherein after a first rough sintering the entire mass resulting is crushed and sintered again, usually with an admixture of a small percentage of new material, both sulfide and oxide. This was not adopted all at once. It probably developed from the practice of screening the sinter and returning the fines to the charge for resintering. As the sinter from fine concentrates began to get weaker and weaker the proportion of these fines became greater and greater until finally some one hit upon the idea of returning the entire first sinter for a second sintering, adding just enough new sulfides to bring the sulfur content up to a point where it would ignite again. It was found that the "pop corn" articles from the first roast were effective in opening the charge for the passage of air. The second sintering also brought the sulfur down in the final product to about half of what it was in the first sinter. The lumps of sticky concentrate which did not sinter the first time over were at least dried so that they mixed better with the second charge.

As observed in the Salt Lake Valley this second or double sinter is a decidedly better product than that which was obtained when fine concentrates were first sintered. It is still rather weak as compared to that made from crude sulfides. Two materials have from time to time been added to the charges to strengthen the sinter, namely crushed or granulated slag, and matte. Although these materials undeniably do strengthen the sinter, their use is not always an ultimate economy.

Reduction of Double Sinter

Reduction of the double sinter in the blast furnace seems a little more difficult as fairly high percentages of coke are used together with considerable scrap iron. The three smelters are using on the average about 13 per cent. of a by-product coke running 85 to 87.5 per cent. fixed carbon. At times the amount used is as low as 12 per cent., or less, and occasionally up to 14 per cent. or more. The amount of scrap iron used on the charge has considerable influence on the amount of coke found necessary for good reduction.

High blast pressures prevail generally. In fact, the Murray plant uses pressures which are reported as reaching at times heights twice as great as those considered safe a few years ago. Tooele has been using a blast pressure of 44 oz. per sq. in. and Midvale about 48 to 52 oz., measured at the blowers. Murray operates on the principle of a fixed amount of air per minute letting the blast pressure go where it will, within bounds. This is somewhat the practice at Midvale although there the maximum is limited to 52 oz. At Murray, it is stated, it has gone to 60 and 70 oz., and even more, momentarily. This is quite an innovation when compared to the old practice of keeping the pressure fairly uniform, regulating the quantity to so maintain it. Ten years ago the blast pressures were more commonly in the neighborhood of 36 to 42 ounces. Piston blowers are used to obtain these pressures, cycloidal blowers having too great a slip.

The logic of this practice is obvious. With a charge composed mainly of sinter, and that weak and friable though porous, the blast-furnace charge column now has a multitude of small passages for the blast instead of the comparatively smaller number of larger passages existing in the days of strong sinter in large cakes. While the actual amount of air per ton of charge is probably no greater, if not less, than in former days nevertheless the resistance is very much greater, hence the higher pressures.

These higher blast pressures although difficult to handle, together with the higher percentages of lead on the charges, have naturally resulted in fast running and new records in tonnages per square foot of hearth area. Without taking into account the so-called "concentration" charges for resmelting matte, which always ran extremely fast, the present rate of smelting puts through 5 to 6 tons of charge exclusive of coke, per square foot of hearth area per 24 hr. Five tons is about the general average, day in day out, without allowance for lost time. Naturally at times when the furnaces have become well "crusted up" and approach the ends of their campaigns the rate is not so good. On the whole, however, there has been a gain in speed. This, plus the higher percentage of lead on the charges, has in turn brought about a rapid running of bullion from the lead wells. The furnaces need close watching for if they do start channeling or blowing through the effect is rather startling. Nevertheless they run surprisingly quietly with the high blasts.

Roasting Equipment

At the Tooele smelter of the International Smelting Co. all lead plant roasting is at present done on 10 Dwight-Lloyd machines, four of which are used for the first or rough roast sinter and six for the finished sinter.

The proportioning of the charges is effected in the conventional manner by individual feeders from the respective bins, all discharging onto one common belt conveyor. Most of these feeders are of the revolving disk, or modified Challenge, type. For sticky concentrates, however, rubber belt feeders have been installed under separate auxiliary hoppers. Concentrates drop into these hoppers from the overhead bins. By this means an opening is left between the hoppers and the bins through which a pokebar can be manipulated to bring down the concentrates when they hang up in the bins above. In addition this pokebar is a piece of pipe attached at one end, through a valve, to a length of air hose, making it possible to use compressed air to further assist in dislodging hung-up material. This whole arrangement is quite helpful. The first conveyor collecting material from the feeders discharges into a large mixer. This is like a very flat inverted frustrum of a cone. Four arms driven by bevel gears and having plows mounted in them turn the charges over and work them into the central discharge hole through which they drop on to the conveyor belt leading up into the sintering building proper. A third conveyor belt distributes the charges into the respective machine hoppers by means of a traveling tripper. From these hoppers the material is fed onto the machines by adjustable revolving disk feeders. A reciprocating swinging chute distributes the charge uniformly across the width of the sintering machine pallets. This same device is used at the Midvale plant of the United States Smelting, Refining & Mining Co.

For ignition all three of the valley smelters use a furnace fired with coke breeze and supplied with a gentle blast of air. The uptake of this furnace makes a complete 180° turn, discharging the gases of combustion downward on the moving bed of charge on the sintering machine. This furnace has proved to be a great help in getting a good ignition of the charges, being in fact much better than the oil-fired muffles originally used.

The entire first sinter, fines and all, is taken to the sulfide mill in railroad cars and crushed down to ¼ in. to ⅜ in. size. After this crushing it forms about 70 per cent. of the second or finishing sinter, the remaining constituents of which are about 10 per cent. siliceous diluent, 10 per cent. crude sulfides and 10 per cent. miscellaneous. On discharging from the machines the sinter drops over a grizzly with ⅜-in. openings into railroad cars for transport to the blast-furnace charge bins. The return fines through the grizzlies go back to the first or rough sinter where they are of benefit in opening up a charge which would be pretty dense otherwise, composed as it is of flotation concentrates, both oxide and sulfides, dust from the Cottrell precipitator, fine diluent, and so forth. The first sinter charges run about 12½ to 13 per cent. sulfur and are roasted down to about 6 per cent., while the finishing charges run about 9½ per cent. sulfur and are roasted to as low as 1.7 per cent. The importance of getting the finished roast as low as possible in sulfur so as to keep the matte fall from the blast furnaces low is self-evident. The ores received at the valley smelters generally have present appreciable percentages of copper so it is not feasible to eliminate the matte fall entirely as is done in some other districts.

PRACTICE AT MURRAY

At the Murray plant of the American Smelting & Refining Co. three Dwight-Lloyd machines are engaged on rough presintering and four on finish sintering. The entire product of the first three machines falls on a chain type pan conveyor which carries it to a pair of Linkbelt toothed rolls, from which the crushed sinter is conveyed to a pair of straight face rolls. After rolling to about ⅜-in. size or smaller the product goes to bins from which it is fed out as the principal ingredient of the second sinter. Some of this first sinter is used also as diluent on new charges of first sinter. Belt feeders from the respective bins discharging onto one main conveyor are used to proportion the charges which are conveyed to conical hoppers over the sintering machines. Under these hoppers are horizontally revolving tables acting as both feeders and mixers through the action of stationary plows which turn the material over and work it to the edges of the plates and over down onto specially shaped chutes to spread it over the pallets. At an intermediate point in the conveyor system, where direction changes, one conveyor drops the charges through a Steadman disintegrator. This machine, originally installed to break up lumps of concentrates, has proved to be also an excellent mixer. Its action plus that of the horizontal mixer tables puts the charges on the machines in a condition approaching the ideal state of thorough mixture.

Multiple-hearth roasters of the Wedge type are used to preroast high-sulfur concentrates, such as pyrite, before submitting them to the sintering treatment. These iron concentrates are particularly troublesome to sinter, making weak cakes unless mixed in the proper proportions with other material of better sintering qualities. Dusting was particularly troublesome on these roasters until the adoption of a mechanism for moistening the calcines as they were discharged.

One detail of practice at Murray which has been helpful in the handling of the troublesome flotation product is the manner in which it is bedded with crushed crude sulfides in the unloading bins before transport to the charge bins of the sintering plant proper. By this means the drier and coarser crushed crude sulfides add their characteristics to the wetter and finer flotation concentrates, resulting in a mixture which can be carried on the conveyor belts to the charge bins and out on the feeder belts without too much trouble. However, even this expedient does not insure against the appearance of lumps of concentrate which would cause much difficulty were they not broken down by the Steadman disintegrator previously mentioned.

MIDVALE

At the Midvale plant of the United States Smelting, Refining & Mining Co. there are one McDougal and five Wedge furnaces and six sintering machines. High-sulfur concentrates, such as pyrite, are given a preroast on the former, the sulfur being brought down to around 12 to 18 per cent. With the fine concentrates are mixed certain plant by-products such as speiss and other miscellaneous materials. A sort of rough bedding together of these materials somewhat as at Murray lessens the difficulties of keeping the fine material moving. Most of these furnaces have belt feeders, but feeders of the apron type such as installed on the McDougal furnace are more positive in their action on fine concentrates. Regardless of what type of feeder may be used close watch of the bins must be maintained to guard against hang-ups. The calcines were formerly very troublesome on account of the dust. This difficulty was overcome by taking off a spray from the water-cooling system of the revolving arms on the bottom hearth, thus making this a moistening and mixing floor rather than a roasting hearth. The calcines treated in this manner cause no further special difficulty in handling beyond that inherent in the poor sintering qualities of high-iron concentrates.

From the calcine bins into which this material is carried by conveyor belt from the multiple-hearth roasters it is transported in V-body tilting cars to the charge bins of the sintering plant. The same cars also haul the other ingredients of the sintering charges from the sampling mill reject bins where they are unloaded. The siliceous fines used for diluent are ordinarily screened from the oxidized ores as they pass through the sampling mill and are distributed by a traveling belt tripper into composite beds in the sample mill bins, beneath which run the V-body cars. The charge bins at the sintering plant therefore draw their supplies from two sources, the preroast calcine bins and the sample mill reject bins. Belt feeders under the respective bins discharge onto one common belt in the conventional manner, which belt delivers the proportioned charges to the bins over the sintering machines. A weighted roller riding on this main conveyor at a point where there are three closely spaced straight idler rollers underneath gives some assistance in breaking down lumps. From the sintering machine bins the charges are drawn out by rubber belt feeders and dropped into cylindrical revolving barrel mixers, 20-in. dia. by 5 ft. long. These mixers, driven through bevel gears, slope downward toward their discharge ends at about 2⅝ in. per ft. Interior lifting rabbles were tried but finally discarded. These mixers, which were a gradual evolution after grievous experiences with other types, have done quite well although they do not insure breaking down of lumps as may be said of practically any type other than the definitely disintegrating ones.

Midvale has like the other two plants been compelled to adopt the double sintering plan. The details of practice are not set hard and fast, varying with the materials to be dealt with. Grizzlies in the discharge chutes are not used at this writing. The first sinter, discharged into railroad cars, is hauled to the sampling mills and crushed to about ¼ to ⅜ in., and then brought back to the charge bins to be proportioned out as the principal constituent of the second or finishing sinter. It is frequently used over again, in small part, as diluent for a new first sinter. Sulfur in the first sinter charge runs about 12 to 13 per cent., being roasted down to 6 or 7 per cent. The second sinter charge runs about 9 per cent. and is roasted down to about 3 to 4 per cent. Discharged into railroad cars without screening it is hauled to the blast-furnace charge bins.

The above outline of the practice of the Salt Lake Valley smelters in sintering is scarcely adequate to give a full picture of the difficulties encountered in handling fine ores. The greatest change in practice and undoubtedly the most helpful is the double roast or sinter. But the fact remains that the concentrates have to be roasted two, three or more times before they are fit for good reduction in the blast furnaces.

Direct reduction of high-grade concentrates with scrap iron in reverberatory furnaces has been discussed but does not as yet seem quite practicable.

The very high grade of some of the flotation concentrates has directed attention to the possibilities of reduction on the Scotch hearth. However, for this work the grade must be maintained at 70 per cent. and better, which is generally not regularly possible. Furthermore, some experimental work along this line appeared to indicate that troublesome dust problems would occur.

If a stronger final sinter could be obtained a lower blast could be used. Perhaps a saving feature in connection with the smaller sinter obtained is the small by-product coke now used. The two mix together fairly well making a sort of thick porous mat which takes air more uniformly through the general mass than the old charges of solid ore and large coke, although at the expense of high pressure and the liability to burst through in case of any uneven charging.

Moisture in Concentrates

A rather curious element in influencing the degree of difficulty encountered in handling flotation concentrates is the percentage of moisture in them. A difference of 2 or 3 per cent. makes a surprising difference in the workability of the material. The more closely this can be controlled the more easily can the sintering charges be handled. Mills with inadequate filtering capacities send in wet, slimy material which is a veritable bugbear. Some well-filtered concentrates move surprisingly well. At the other extreme are concentrates which have been fairly well filtered and then given an air-drying under a hot sun, resulting in baking the material together in lumps which actually required crushing before they could be sintered. Since it must be admitted that it is difficult to control the moisture at the mills the smelters assumed the duty of taking everything as it came. One plant reports that by carefully regulating the moisture in the charges as compounded for the sintering machines considerable improvement in the strength of the product has been effected. From 7½ to 8 per cent. moisture on the first sinter and from 5.8 to 6 per cent. on the finished sinter appear to give the best results.

Swing-hammer pulverizers, squirrel-cage disintegrators, toothed rolls, single-roll crushers and other devices have been tried and some are still in use for crushing and mixing. Manufacturers of such machinery have given the problem study and rendered valuable assistance. Yet it must be clear that the present-day roasting plants constitute astonishing assemblages of machinery with all their feeders, conveyors, crushers, mixers, etc. Simplification would be most desirable but so far it does not appear to be in sight for plants that would avoid that enemy of good reduction in the blast furnaces, fine unroasted sulfides.

Yet continual progress is being made as the result of the unremitting efforts of the metallurgists to cope with the constantly changing conditions. A hopeful sign for the future is the mutually helpful attitude displayed toward one another. Realization that this spirit exists emboldens one to seek criticism and suggestions for further improvements. For example, what experiences have any of the readers of this paper had in conveying hot sinter? What are good feeders and good bin shapes for sticky concentrates? What experiences have there been with the different types of crushers for sinter? How are flotation concentrates best sampled? What is done to allay dusting and fuming, aside from water spraying? What are the types of mixers which have been found satisfactory? If responses to such questions and allied ones which may suggest themselves are forthcoming then the writing of this paper will have been well worth while.

DISCUSSION

R. L. Lloyd, New York, N. Y. (written discussion).—Mr. Anderson has given us an excellent and very helpful discussion of a most troublesome problem. It divides itself naturally into two subjects which are from the cost per ton point of view nearly equal in importance. The first subject has to do with the chemistry of the process from the metallurgical point of view and this chemistry is somewhat affected by the mechanical condition in which the product must be furnished to the furnaces. The second subject is the mechanical handling of the modern extremely fine, slimy and gummy concentrates which now constitute the lead smelters' main supply of lead-bearing materials.

Discussing the chemistry of the process, the main object, of course, is to furnish to the blast furnaces a desulfurized material, incidentally with the least possible loss in metals in the process of desulfurization. It was early recognized that the sintering process and the apparatus which may be used in carrying it out were limited to a certain fuel range inside of which the best and cheapest work could be accomplished, therefore roasters were not displaced in the case of the highest sulfide ores. On the other hand, the introduction of sintering permitted roasting itself to be carried on in its own best range, so that roasting could be done much more cheaply than previously.

The economic fuel range of sintering comes in below the economic fuel range of roasting, while the temperature range of sintering is much above the temperature range of roasting, so the invention of sintering put into the hands of the metallurgist an additional tool and made it possible to attain a final result through the best range of roasting and the best range of sintering, that was a much better product chemically and physically than the roasters could make alone; indeed, the product was immensely superior to anything that had been previously known; also, the combined procedure was cheaper than the roasting process alone, which heretofore had been pushed beyond its range. In other words, the net result was a new metallurgical product, and whereas the product before had been at the best a hit or miss sort of agglomerate varying from stony slag to dusty fines, we now have the uniformly porous and chemically constant material which we know as sinter. It is to the physical condition of this product, sinter, that I refer when I say that the chemistry of the process is itself concerned with the physical nature of the product. Originally sintering started out to make this product in one bite, so to speak, and while it often still does so, better and cheaper results can be obtained in many cases by taking two bites. To take two bites requires, to all intents and purposes, the same amount and kind of equipment that is required to take one bite, except that there has to be an intermediate crushing between the bites; and so grew up double sintering, which is in its way analogous to stage pumping as compared to the single lift.

Double sintering had been approximated by the addition of return fines, even sometimes crushed sinter, before true double sintering was done at the works of the St. Joseph Lead Co. in Missouri.

The conditions which now prevail so far as the sintering itself is concerned would have prevailed in any event, even though the flotation process had not changed the physical character of the lead smeltermen's ores.

The second part into which Mr. Anderson's paper naturally divides itself deals with the mechanics of the handling of the extremely fine flotation concentrates. This problem is not yet solved. Mr. Anderson calls attentionto bedding methods, which are a partial solution, though not always available. He also calls attention to mechanical methods, the chief feature of which is to have the receiving bins discharge largely by main strength into auxiliary bins from which the concentrates are handled by feeding devices, these devices being possible on small bins although they would not work at all on large bins. The small bins act as rectifiers between the large bins and the machines or mixing devices. It is unquestionably true that the mechanics of the handling of present-day smelter input, from a dollars and cents point of view and the possibility of savings, is a problem as great as or greater than the sintering process itself, even when account is taken of the mixing back with the original concentrates of coarse and dryer materials such as flux, diluent or even returned or recrushed material from the second stage of the operation.

I believe that the mechanical handling of flotation concentrates is working toward a radical change, although not by the introduction of any new or untried machinery or apparatus. I believe that flotation concentrates should be, and shortly will be, sluiced from the cars into thickening tanks, from which they will be delivered into the sintering flow sheet directly from filters, feeding onto belts. The filters themselves will be controllable as to speed and act as feeders into the main stream for the sintering machines. The current supply of concentrates will be carried at all times in the thickening tanks. Such procedure will prevent balling of the concentrates and eliminate all bin troubles and facilitate mixing to a degree not now approached. This, I know, presupposes a considerable amount of costly equipment, but it will materially aid the general smelter operations, including sampling. Since most if not all smelters have sources at which waste heat is available, the matter of the winter handling of thickening tanks is comparatively simple.

Mr. Anderson has handled very ably the question of mixers, disintegrators and feeders, and it may be said unequivocally that proper sintering demands proper mixing beforehand. The mixing of flotation concentrates with other materials is complicated by their tendency to lump, and this tendency to lump and consolidate is quite disproportionately accentuated if the concentrates are binned, hence the need for disintegrators and other devices in the first step. The best of the mixing devices, as distinguished from disintegrators, so far as I know, is the rotary pug-mill mixer which has recently appeared. The squirrel-cage type of disintegrator is nearly standard where such apparatus is indicated at all. So far as feeders are concerned, and for any kind of material that will discharge readily from bins, the belt feeder with adjustable gate is the best all-round device, and for distribution on the sintering machine nothing has yet appeared that excels the swinging spout.

It is undoubtedly true that the present product made from flotation concentrates is more friable than the product made from crude ores and gravity concentrates, which were less free from gangue than are the flotation concentrates. The reason for the present greater friability of sinter is precisely the same as for the furnaces now making a great deal less slag per unit of lead produced than they previously did. It was the presence of greater quantities of silica and other gangue materials in the older practice that made the sinter harder and firmer than it is at present. So again, greater production of lead per unit of furnace area has brought with it its more or less counterbalancing difficulties, but the net result, of course, is a gain.

Unfortunately, all the lead smelters of note in our country were designed and built before the coming of flotation concentrates, and most of the handling difficulties arise from this fact. It is safe to say that no metallurgist designing a new smelter at this moment would follow closely the old handling designs, when gravity concentrates and crude ore were the source of supply. Some of the modern smelters were designed after the birth of sintering, but even the older smelters had no difficulty in interpolating a sintering plant which became a unit of itself.

This interpolation was in the nature of an addition to equipment. It is a different matter to interpolate a handling plant which involves the total or partial abandonment of present bin equipment, nevertheless I am strongly of the opinion that flotation concentrates should be handled along the lines which I have indicated and that the result of such handling will be reduced costs all down the line, from the reception of the concentrates to the finished sinter.

H. W. Hixon, New York, N. Y.—The plant at East Helena, while I had charge of it some years ago, consisted of what we called flat-top furnaces, as distinguished from those in which the stack goes up above the charge floor; also a goose-neck to the dust chamber. There seemed to be an epidemic of building flat-top furnaces. These furnaces get hot behind the feed thimble and the lead losses were high. The reason is simple when you understand it. If you feed through a thimble that is smaller than the shaft, there is always a spilling action toward the walls and the natural result is that the coarse pieces go to the wall, the fine go down in the center of the furnace and the blast comes up where the resistance is least and results in over fire between the wall and the feed thimble. We would have to take out the thimble and feed the furnace with a shovel and endure the smoke. This was later changed to an automatic feed car system.

The furnaces were about 20 ft., and subtracting the length of the feed thimble, the effective charge column was about 16. We had to smelt with 10 per cent. lead on the charge.

When charge is sintered the smelting is a different proposition from smelting unsintered ores, so I cannot say that there is any basis for comparison of the two things. One seemed to require a higher charge column than the other. We used blast pressures at East Helena at that time up to 40 oz. in connection with a 16-ft. smelting column, and considerable zinc going into the slag. Of course we smelted ores in those days, losing all the zinc in the slag. It would be a metallurgical crime to do that now.

C. R. Hayward, Cambridge, Mass.—I am impressed by what has been said in regard to the lower height of the charges. I have been convinced from observation in the plants and even in running our little laboratory blast furnace that that is the correct way to operate blast furnaces. We naturally think that it is advisable to run with higher charges in order to get proper preheating, but in our little laboratory furnace we have found that probably lower charges will give more satisfactory results. This is particularly true with some high-porosity sinter that is now being produced. It is possible to get thorough reducing conditions in a short passage of the gases. Earlier in the meeting something was said in regard to the size of the coke. I thought it was generally understood now that it is desirable to have the coke in rather large lumps so that it will be present at the tuyeres and insure thorough reducing conditions right where they are needed. Small coke always burns out toward the top of the furnace and does not leave sufficient coke in the charge at the tuyere zone to produce the high temperature needed there, and the strong reducing conditions, particularly a large amount of CO gas starting at the tuyeres.

R. C. Canby, Wallingford, Conn.—The Dwight-Lloyd people had an experience which I had the opportunity of observing, which suggested to me the term "predigested furnace feed." That may explain the possibility of the low shaft charge. There is possibly more reduction in sintering than is usually supposed. This experiment was the treatment by sintering of roasted pyrite containing ferric oxide. After sintering, it was possible more readily to smelt it in the reverberatory furnace.

E. L. Jorgensen, Irvington, N. J. (written discussion).—I believe that the value of this kind of paper would be still greater if in addition to the data on blast pressure were given information about the volume of air used in terms of cubic feet per minute and per ton of charge as well as the dimensions of the furnaces, the percentage of coke and the analysis of the gas leaving the furnace.

A recording instrument for air volume should be considered at least as important as the recording pressure gage on any blast furnace. It is not necessary to invest in any of the costly flow meters. An inexpensive standard recording differential pressure gage will do if equipped with some simple home-made safeguards. As these safeguards are not considered by the manufacturers, I must go into detail.

The recording differential pressure gage is made exactly like the usual recording pressure gage, except that the housing of the instrument is made airtight and is fitted with an airtight door. By inserting a restriction or orifice in the airline, a pressure differential is created between the two sides of the restriction. By connecting the higher pressure side to the gage and the lower pressure side to the housing of the gage, the differential pressure corresponding to any volume for the particular size and shape of restriction is recorded.

These gages are naturally designed for a much lower range than the actual pressures in the air mains.

When placing the instrument in service it is practically impossible to admit air so evenly to both sides of the instrument that its range is not exceeded. If air is admitted too fast to the housing, the range below zero is exceeded and if admitted too slowly the maximum range is exceeded. In either case the instrument will be damaged and must go back to the factory for repairs. Our two first attempts resulted in such failure. Therefore we designed our own safeguards against recurrence.

Since the metal bellows in the gage has very small volume, the line leading to it is provided with an air chamber, made of pipe, of approximately the same volume as the instrument housing. This chamber is located close to the gage, and serves to equalize the time required for the air in the bellows and the air in the housing to reach stable conditions when cutting the gage in or out of service.

Just ahead of this chamber is a cross-over line, connecting the two ¼-in. pipes from the air main. This cross-over pipe is provided with a shut-off valve, which when closed renders the cross-over inoperative, and also with a blow-off valve connected in by a tee in the cross-over line. This blow-off valve may be placed either side of the shut-off valve. With both of these valves open, no appreciable pressure can reach either side of the gage, whether one or both of the main control valves happen to be open. It is this feature which insures the safety of the gage mechanism.

Just ahead of the cross-over line are the main control valves for cutting the gage in or out. There is one control valve in each of the two ¼-in. lines that lead from the air main to the gage.

When ready to cut the gage in, the shut-off valve and the blow-off valve must both be open, in the cross-over line, before turning on the main control valves. After these are on, the blow-off valve is closed first, and then the shut-off valve. This applies the pressure simultaneously to both sides of the bellows. When ready to cut the gage out, the valves may be manipulated in any of several different sequences, but damage to the instrument will always be prevented if opening the shut-off valve in the cross-over line is made the first step. As a further safeguard the instrument and the valves were placed in a locked cabinet to prevent unauthorized persons from touching the valves.

At the Irvington Smelting & Refining Works, our air pressures may vary from 4 to 20 oz., depending on the nature of the furnace charge and the height of the column in the furnace. Positive knowledge of the volume of air is invaluable to predetermined performance. If air volume is maintained at a predetermined amount one unknown factor is eliminated in diagnosing furnace trouble.

Gas analysis is of great value in fixing the amount of coke to be used. I am informed that at the Murray smelter of the American Smelting and Refining Works air volumers are regulated by gas analyses, 15 per cent. CO being considered the best point. At Irvington we run 13 to 14 per cent. CO. In case of furnace trouble and irregularity we first of all make a gas analysis and generally we find a subnormal CO due to shortage of coke. If, however, the CO content of the gas is normal we can start to diagnose the trouble with much more certainty of guessing right.

I believe these points of air volumes and gas analysis are the most important and most underestimated factors in blast-furnace operation.

Just a few words about coke. I believe that in a high column coke will disintegrate more than in a low column. The iron blast furnace and the iron foundry cupola both require a coke of great mechanical strength which mean an ash content of 12 to 14 per cent., and also a low reactivity. With a low column a coke with 6 per cent. ash has sufficient mechanical strength, a higher porosity and reactivity. Substitution of the foundry coke with this low-ash coke resulted with use in a material saving in the cost of coke per ton of furnace charge. It also gave smoother furnace operation.

L. D. Anderson (written discussion).—It is gratifying to have the discussion of this paper participated in by men of such standing and attainment in the metallurgy of lead. The fact that they are backed by many years of experience makes their remarks of deep significance.

One important point that has been brought out is with reference to the best blast pressure to be used on blast furnaces smelting sinter. Several smelters are using high blasts and getting good results. May not equally good results be obtained with lower furnace columns and lower blast pressures? The remarks of Messrs. Hayward and Canby are suggestive of thought in this direction.

The Midvale and Murray smelters have done some experimental work with air meters. Some means of accurately measuring the amount of air being supplied each furnace has been the desire of nearly every progressive metallurgist. Difficulty has been experienced, however, in the installation of the meters on account of the fact that the blast mains are not generally arranged to provide for the long straight run of pipe that is necessary to get consistent results. At Midvale it was found that the air meter was not dependable because the run of pipe to it was entirely too short. Some day this may be corrected, and a meter may prove to be a more valuable instrument than it has been thus far.

Large, strong coke is most desirable. Economic conditions have compelled the smelters to use the smaller, weaker by-product coke. It may be that the lead smelters will go through the same experiences as the iron blast furnaces. I recall visiting several iron furnaces in the vicinity of Chicago and Cleveland when they were just beginning to use by-product coke. The men in charge of those furnaces at that time were outspoken in their objections to the kind of coke they had to use. A few years later nothing was heard of the difficulties occasioned by this coke, and upon inquiry I learned that they had become used to the coke, learned how to handle it, and were getting along well. It may be that lower charge columns and lower blast pressure will aid in smelting with smaller, weaker coke.

Lead Alloys for Anodes in Electrolytic Production of Zinc of High Purity

By U. C. Tainton,* A. G. Taylor* and H. P. Ehrlinger,* Kellogg, Idaho

(New York Meeting, February, 1929)

For the last 15 years lead has been the standard material for anodes in electrolytic zinc production and it has been generally accepted that this lead should be as free as possible from impurities. Laist[1] says, "Next in importance to the purity of solution is the purity of the electrodes and tank lining material. The purest lead obtainable should be used for anodes." The same conclusion is expressed by Ingalls[2] and by two German investigators who say that experiments have demonstrated that the purer the lead, the greater is its resistance to corrosion during electrolysis.[3]

Pure lead, however, while the best material hitherto available for anodes, is not entirely free from objection. The plain lead anode gradually disintegrates under electrolysis, so that the life of an anode, say ¼ in. thick, is frequently not more than two years. Part of the lead from the anode finds its way to the cathode, lowering the purity of the deposited zinc and decreasing the hydrogen overvoltage. The rest of the lead that comes from the disintegration of the anode goes into the manganese dioxide which is precipitated in the cells, rendering unsaleable what might otherwise be a valuable by-product. Yet a further disadvantage of plain lead anodes comes from their tendency to bend or buckle during electrolysis, as a result of intercrystalline oxidation. This makes it necessary to use a fairly wide spacing between anode and cathode to avoid short circuits, and thus leads to a higher power consumption than would otherwise be necessary. The power consumption is also affected by the high decomposition voltage at a lead peroxide surface.

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[1] F. Laist and R. B. Caples: The Electrolytic Zinc Process. Handbook of Nonferrous Metallurgy (Liddell). New York, 1926. McGraw-Hill Book Co.

[2] W. R. Ingalls: Metallurgy of Lead. Handbook of Chemical Engineering (Liddell). New York, 1922. McGraw-Hill Book Co.

[3] H. Hock and F. Klawitter: Über das Verhalten von Bleianoden bei der Zinksulfatelektrolyse. *Metal und Erz* (1925) **22,** 377. Abst. in *Chem. Absts.* (1926) 3648 and in *Chem. Zentr.* (1925) **96,** 1784.

This paper describes a research directed towards overcoming some of these disabilities, by the use of lead alloys instead of pure lead. This idea was based upon an earlier observation of one of the authors to the effect that antimonial lead anodes are more resistant to attack than are pure lead anodes when used in certain special types of cell employed for electrolysis of cyanide solutions. The work also owes much to the inspiration of the excellent results achieved by Dr. Colin G. Fink and his co-workers in the development of the copper-silicon-manganese anode for the production of electrolytic copper. It was hoped that by combining other elements with lead it would be possible to produce an anode more resistant to corrosion and having a lower decomposition voltage.

Alloys Investigated and Method of Testing

We decided to concentrate attention on metals that could be made to alloy easily with lead and particularly those that formed more than one oxide. Investigations were made of lead alloyed with the following elements, either singly or in combination: arsenic, antimony, barium, bismuth, calcium, cerium, copper, mercury, silver, thallium, tin.

The tests were designed to reproduce practical conditions as closely as possible. Six small cells of 12-liter capacity, similar in all respects, were made up from sheet lead and arranged for connection in series. In these were placed grooved frames of hard rubber holding anodes and cathodes at fixed spacing. The cathodes were of commercial aluminum sheet, ⅛ in. thick, having a total immersed area of 20 sq. in. The anodes were sheets ⅛ in. thick, cast or rolled from the alloy under investigation, and were drilled with ½-in. holes to duplicate our usual practice. The solution was made up by dissolving an excess of electrolytic zinc in pure sulfuric acid and precipitating out traces of iron with zinc oxide and permanganate. Manganese sulfate was added to give a manganese content of 5 g. per liter.

Electrolysis was carried out at our usual current density of 100 amp. per sq. ft. and a free acid concentration of 220 to 300 g. per liter of H_2SO_4. Air lifts were provided for circulating the solution and water-cooled lead coils for control of temperature. Conditions during electrolysis were kept as uniform as possible in all cells, the aim being to have the composition of the anode the only different factor.

Deposits were stripped at regular intervals, usually 12 hr. Electrolyte was withdrawn as required and replaced by neutral solution to maintain the proper acid and zinc concentration. The factors determined were: current efficiency, terminal voltage, character of zinc deposit, amount of lead in zinc, amount of lead in manganese dioxide precipitate. These were compared with a standard cell containing pure lead anodes and each test was continued until definite indication had been obtained as to whether or not the particular alloy showed any marked difference from ordinary lead.

It was recognized that in carrying out the tests in this way we are dealing with a complicated set of interdependent factors and are not confining the equation to one variable. For instance, in cells having different anodes the manganese in the solution will be precipitated at or on the anode to a greater or less extent, perhaps changing the ohmic resistance, almost certainly affecting the oxygen overvoltage. Also, as the manganese is precipitated, sulfuric acid is set free and the ratio of zinc to acid in the different cells will change. Again, the addition agents, such as glue, in the electrolyte will be adsorbed or chemically changed to a variable extent by the precipitation of the manganese dioxide and the nascent oxygen or ozone from the anode. This changes the surface tension of the solution, which governs the adhesion of the hydrogen bubble to the cathode surface and thus largely controls the smoothness of the deposit. This in turn varies the cathodic current density, and so on.

Because it is not possible in commercial work to isolate these different effects, the test conditions were chosen to reproduce the requirements of actual practice. The object was to determine the final result rather than the means by which it was attained. There remains a wide and interesting field of investigation in the development of the theory to cover the facts here presented.

Alloy Adopted for Use

Within the compass of this paper it is not possible to present more than a few of the voluminous data that have been accumulated in this investigation. While a number of the alloys showed advantages over plain lead, there was one so outstandingly superior that, as soon as it was found, special attention was concentrated on it, leading eventually to its adoption in the large plant. This was Pb-Ag series containing no free silver; *i. e.*, at or below the eutectic of 2.6 per cent. Ag. The account will be confined to this particular alloy and the result of the tests on the other alloys will be presented at a later date.

In commencing the work arbitrary selection was made as to composition of alloy in the light of economic and other factors. We will describe the results from the alloy made up to represent 1 per cent. Ag and 99 per cent. Pb.

In the tabulated results (Tables 1 to 5) current density is, in all cases, 100 amp. per sq. ft.; acid range, 220 to 300 g. per liter H_2SO_4; electrode spacing, 1.25 in., face of anode to face of cathode unless otherwise stated; voltage readings every 30 min.; deposits stripped every 12 hr., sometimes every 24 hr. All figures are averages obtained in 6 to 20 days' continuous operation.

TABLE 1.—*Record of First Fortnight's Work*

	Terminal Voltage	Addition Agent	Current Efficiency	Lead in MnO_2	Lead in Zinc	Character of Deposit
First Week's Operation						
Standard anode, rolled lead, 99.99% Pb.	3.77	Glue 3 lb. per ton Zn	86.15	18.5	0.055	Normal, smooth
Test anode, 1% Ag, 99% Pb.	3.65	Glue 3 lb. per ton Zn	83.5	2.1	0.009	Rather dark, cratered
Second Week's Operation						
Standard anode, rolled lead, 99.99% Pb.	3.82	Glue 3 lb. per ton Zn	85.9	24.5	0.081	Normal, smooth
Test anode, 1% Ag, 99% Pb.	3.63	Glue 3 lb. per ton Zn	82.2	1.42	0.003	Cratered

The indications in Table 1 are definite. We have an alloy anode inferior (under these conditions) in point of current efficiency to the plain lead anode, but superior in point of terminal voltage and resistance to attack. Examination under the microscope of the two anodes showed that the oxidation of the plain lead anode had proceeded to an appreciable depth while the silver-lead anode was practically untouched. This at once suggested that with this alloy buckling under electrolysis would be done away with and it would therefore become possible to use a closer spacing. Table 2 shows the result of closing the current gap to ⅝ inch.

TABLE 2.—*Results with Closer Electrode Spacing*

	Terminal Voltage	Addition Agent	Current Efficiency	Lead in MnO_2	Lead in Zinc	Character of Deposit
Standard anode, rolled lead, 99.99% Pb.	3.7	Glue 3 lb. per ton Zn	85.0	26.0	0.057	Smooth, normal
Test anode, 1% Ag, 99% Pb. Spacing ⅝ in. face to face.	3.32	Glue 3 lb. per ton Zn	81.8	1.72	0.002	Cratered

Here we have a very low terminal voltage for the alloy anode; still the cathode current efficiency is below normal. Evidently the alloy anode in some way changes the solution. In this case there was little difference between the cells in concentration of zinc, acid, and manganese. However, there was considerable difference in the nature of the cathodes:

deposits from the alloy cathode cell were rough and showed the honeycomb character caused by hydrogen bubbles sticking to the cathode. Since the same amount of glue was added in each case, it is probable that the alloy anode in some way destroys or changes the colloid, giving a rough deposit, lowering the current density and so the current efficiency. In confirmation of this it may be mentioned that if the alloy anodes are used for the electrolysis of zinc solution free from manganese an unusually large amount of ozone is given off. Ozone, of course, rapidly oxidizes most organic compounds.

These considerations led to the idea of substituting for glue colloids which would not be decomposed. The first one tried was silicic acid, which was added to the solution in amounts corresponding to 2 lb. of soluble SiO_2 per ton of zinc deposited. Results are shown in Table 3. (Note that the standard cell has now become lead-silver alloy.)

TABLE 3.—*Results with Silicic Acid as Addition Agent*

	Terminal Voltage	Addition Agent	Current Efficiency	Lead in MnO_2	Lead in Zinc	Character of Deposit
Standard anode, 99% Pb, 1% Ag.	3.71	Glue 3 lb. per ton Zn	82.1	2.18	0.0005	Rough, cratered
Test anode, 99% Pb, 1% Ag.	3.66	H_2SiO_3 2 lb. per ton Zn	89.2	1.79	0.0002	Smooth, bright
Standard anode, 99% Pb, 1% Ag.	3.64	SiO_2 2 lb. per ton Zn	85.7	0.6	0.0035	Smooth, bright
Test anode, 99% Pb, 1% Ag.	3.625	SiO_2 4 lb. per ton Zn	84.8	1.1	0.0047	Smooth, bright
Test anode, 99% Pb, 1% Ag.	3.59	SiO_2 6 lb. per ton Zn	85.2	1.4	0.0062	Exceptionally smooth

Finding that the silicic acid addition evidently corrected the trouble with regard to the adhesion of the hydrogen bubbles to the zinc deposit, tests were made to determine what amount would represent the best quantity under these particular conditions. Results are shown also in Table 3.

On the basis of these results, it was decided to install the 1 per cent. silver-lead anodes in the cells of the test plant in place of the plain lead anodes previously used. Each cell in the test plant contained two cathodes taking 400 amp. each, so that results obtained there could be directly translated into large-scale practice. On account of the non-buckling property of the alloy, the anodes were set much closer to the cathodes than before. The previous spacing of 1¼ in. was reduced to ⅝ in. Silicic acid was added to the solution, as had been done in the small cells.

The larger scale operation fully confirmed the findings of the small cells. The average voltage dropped from 3.7 to 3.4, occasionally going as low as 3.2. The zinc deposit began to show complete absence of lead when tested by the methods hitherto used. The zinc became, as time went on, more and more difficult to dissolve in acids so that eventually even the solution of the sample in hydrochloric acid for cadmium determination took upwards of a week.

Tests were made with alloys containing greater amounts of silver, up to as much as 25 per cent. It was not, of course, expected that use of anodes cast of this alloy would be practicable in large-scale work; the expense would be excessive. The idea was that we might form a thin layer of high-silver alloy on each side of a plain or hard lead base, either by rolling a billet made up in this way or by plating a layer of silver on the lead and causing it to alloy with the lead by heating. However, in all cases where the quantity of silver exceeded the eutectic of 2.6 per cent. the zinc deposits showed lowered efficiency and sometimes serious re-solution. Frequently it was impossible to get any deposit after the first hour or so. Evidently the free silver from the alloy dissolves in the electrolyte and acts towards the zinc deposit in the same way as would any other noble metal.

At or below the eutectic point this effect is not strongly noticeable. However, the extra cost of increasing the silver content above 1 per cent. does not appear to be justified by commensurate returns in power economy and quality of zinc, except possibly under special conditions. The alloy below 1 per cent. silver appears to disintegrate a little more rapidly and to increase the amount of lead in the deposits. A proper test of such a point requires several months of continuous running and the investigations are still proceeding.

Effect of Various Addition Agents

After some time the zinc deposited in the test plant from the anodes containing 1 per cent. silver and with only silicic acid as addition agent began to show considerable re-solution and loss of efficiency. Examination disclosed the fact that the deposit was slightly porous and that the acid was finding its way behind the deposit and corroding it from the starting sheet side. To remedy this we tried the addition of an organic colloid, gum arabic, which we knew from past experience would remedy this particular condition of porosity. It had the desired effect and since then it has been our practice to associate with the silicic acid agent a small quantity of an organic colloid, which renders the cell more stable in case of impurities in the electrolyte.

Several organic colloids were tested in the small cells, such as starch, gum tragacanth, gum catechu, and agar agar. With the exception of the agar, none of these gave results as good as those obtained with the silicic-arabic combination. Table 4 gives the result of a series of tests varying the quantity of gum arabic added and also shows results with some other addition agents, for comparison.

The results in Table 4, as also those in the two preceding tables, were obtained with solution derived from the leaching of ores in the test plant. This is probably responsible for the rather erratic current efficiency figures when different tables are compared. However, since the same solution was used for each individual set of tests the results are comparable among themselves.

TABLE 4.—*Tests with Various Addition Agents*

Test Anode, 1 per cent. Ag, 99 per cent. Pb

Terminal Voltage	Addition Agent	Current Efficiency	Lead in MnO_2	Lead in Zinc	Character of Deposit
3.57	SiO_2, 2 lb., gum arabic, 1 lb. per ton Zn	80.5	1.5	0.0055	Gray, slightly cratered
3.54	SiO_2, 2 lb., gum arabic, 2 lb. per ton Zn	83.0	1.4	0.0056	Gray, fairly smooth
3.52	SiO_2, 2 lb., gum arabic, 4 lb. per ton Zn	81.0	1.5	0.0056	Gray, fairly smooth
3.61	SiO_3, 2 lb., gum arabic, 2 lb. per ton Zn	86.0	Not determined	Not determined	Smooth, bright
3.62	SiO_2, 2 lb., agar, 0.5 lb. per ton Zn	87.7	Not determined	Not determined	Dull, extremely tough
3.74	SiO_2, 2 lb., glue, 3 lb. per ton Zn	78.0	Not determined	Not determined	Dark, cratered

Because there were other more pressing matters to be attended to, the testing of addition agents was not carried further, but a standard of 2 lb. SiO_2 and 2 lb. gum arabic was selected and used continuously in the test plant. As continuous operation and steady conditions were more easily obtained in the larger plant, the current efficiency and voltage figures were considerably better than in the test cells. The average figures for terminal voltage and current efficiency in the large cells over one month's operation under these conditions were as follows: terminal voltage, 3.37; current efficiency, 91.5; lead in zinc, 0.004 per cent. These conditions were selected for use in starting up the large plant, which has recently gone into commission.

TESTS ON TERNARY ALLOYS

Because of indications obtained from work with other alloys, tests were made with ternary alloys of lead, silver, and another metal. Combinations were made of lead with silver-tin, silver-copper, silver-mercury, silver-arsenic, and silver-thallium. We are still investigating the matter. There appears to be some advantage in the lead-silver-arsenic and the lead-silver-tin combinations (Table 5). The lead-silver-tin anodes give exceptionally smooth deposits with consequent high efficiency while the lead-silver-arsenic anodes put very little lead in the zinc and manganese dioxide.

TABLE 5.—*Tests with Different Anodes*

	Terminal Voltage	Addition Agent	Current Efficiency	Lead in MnO_2	Lead in Zinc	Character of Deposit
Standard anode, 1% Ag, 99% Pb.	3.61	SiO_2, 2 lb., gum arabic, 2 lb. per ton Zn	86.4	0.26	0.0007	Normal, slightly cratered
Test anode, 1% Ag, 0.5% Sn, 98.5% Pb.	3.57	SiO_2, 2 lb., gum arabic, 2 lb. per ton Zn	87.6	0.18	0.0005	Exceptionally smooth, satin sheen
Test anode, 1% Ag, 0.2% As, 98.8% Pb.	3.60	SiO_2, 2 lb., gum arabic, 2 lb. per ton Zn	85.8	0.06	0.0003	Dull, slightly cratered

OPERATION OF LARGE PLANT

In the construction of the large plant, the anodes were made of an alloy containing 1 per cent. silver and a small quantity of arsenic. This alloy is considerably harder than plain lead and therefore better resists mechanical handling. The spacing was 5/8 in. from face of anode to face of cathode. Anodes were cast 1/8 in. thick, with holes, so that 40 per cent. of the area was removed. Two of these anodes are used between each pair of cathodes, the cathodes being 4 in. apart, center to center. So close a spacing would not have been practicable with single plate anodes, as the volume of the solution between the electrodes would have been insufficient to supply the zinc required at the cathode without extremely rapid circulation. With the construction described, however, the volume of solution undergoing electrolysis between each pair of cathodes is no less than usual and it can freely circulate through openings in the anodes. Arrangements are provided in the cells for maintaining proper parallelism of all electrode faces, so as to insure uniform distribution of current density. The voltage in the large plant has averaged

3.35 volts per cell, occasionally going as low as 3.18 at the maximum acid concentration. The current efficiency varied considerably at the start owing to irregularities in the supply of neutral solution, the average being around 87 per cent. and occasionally reaching 94 per cent. Tests are still being continued to determine whether any better combination of addition agents can be found.

In the matter of purity of zinc, the operation of the large plant has exceeded the expectations entertained as a result of the small tests. Analysis by Ledoux & Co., over an average sample of zinc produced during one month's operation, shows the following: cadmium, 0.0005 per cent.; lead, 0.0015; copper, 0.0008; iron, 0.0004; zinc, by difference, 99.9968 per cent. Metals other than cadmium, lead, copper, and iron not found.

It is perhaps only fair to say that our own analysis over the same period shows a somewhat lower purity, namely 99.994 per cent., and this is illustrative of the extreme difficulty attaching to accurate determinations of minute amounts of impurities, especially cadmium.

The properties of metal of this degree of purity are quite surprisingly different from those of ordinary high-grade zinc. It is very malleable and can be rolled indefinitely without annealing. It can be beaten out into extremely thin sheets like gold leaf—as thin as one sixteen thousandth of an inch. It can be put through drawing, stamping, and extrusion operations of a severity hitherto beyond the range of commercial zinc metal. It is almost unaffected by sulfuric acid and only slowly by hydrochloric. It possesses great resistance to atmospheric oxidation and if not exposed to electrolytes appears to keep bright indefinitely. It is finding a market among manufacturers of brass, zinc-alloy die castings, and stamped products, and in these fields is producing results not heretofore attained.

DISCUSSION

C. S. Witherell, New York, N. Y.—Mr. Tainton, relative to your electrolytic zinc of exceptional purity, will you kindly tell us something about its resistance to corrosion and about the difficulty of dissolving it in sulfuric acid?

U. C. Tainton.—When we were making electrolytic zinc about five years ago we used to make some determinations by dissolving the zinc in sulfuric acid, but after we began to use the lead-silver anodes, the solution in sulfuric acid got slower and slower until finally it took about a month, so we had to give up that method. We then went to solution in hydrochloric acid, but as the zinc got purer, that test began to take from several days to a week to dissolve the sample. Now we use nitric acid entirely for the solution of the zinc. The slow rate of acid attack is an important point because the electrolytic operation is sometimes subject to power interruption. In the old days, it was considered necessary to get the cathodes out of the solution cells as soon as the power went off and it was exceedingly difficult sometimes. With this pure metal, there seems to be no tendency to redissolve; we have had power interruptions as long as 3 hr. without any trouble at all.

H. R. HANLEY, Missouri School of Mines, Rolla, Mo.—I notice the terminal voltage was greatly reduced. This, of course, is an important subject economically and technically. When additional agents are added, they have some effect either on the cathodic voltages, and anodic voltage or the ohmic resistance. Did the addition of silver have such an effect on the reduction of the single potential of the anode?

U. C. TAINTON.—The reduction in terminal voltage was observed in cells that are identical in every respect except in composition of the anode. We did not make any direct determinations of the potential of the different electrodes because, for reasons that I have mentioned in the paper, the changing of any one variable affects so many other variables that it is almost impossible to isolate the different factors. We were concerned in this work with the development of a practical operation and we had to meet certain conditions as regards the composition of solution, so we governed our tests purely from that standpoint.

O. P. WATTS, Madison, Wis. (written discussion).—The electrolytic recovery of zinc is an excellent illustration of the usual time lag between the discovery of some law of nature and its application in industry. The variation in the potential necessary for the deposition of hydrogen on different metals was known many years before its application to electrolytic zinc recovery made this process a commercial possibility. It was only when zinc solutions were freed from metals on which the discharge potential of hydrogen is less than the potential of zinc that deposition of zinc from moderately acid solutions became possible. The next forward step was the demonstration of the possibility of depositing zinc from highly acid solutions, even more acid than is used for refining copper, which was accomplished by the combination of extremely high current density with the use of an addition agent. This combination results in covering with zinc those traces of other metals which are still contained in the electrolyte after the customary purification, which are deposited along with the zinc, and at lower current density serve as points on which hydrogen is liberated instead of zinc.

It has long been known that the discharge potential of oxygen, like that of hydrogen, varies on different metals, but it has remained for Mr. Tainton to apply this principle usefully in lowering the polarization at insoluble anodes. This is an important improvement in the electrolytic recovery of metals from solution, though unfortunately its scope is limited by the solubility of most metals as anode.

C. G. FINK, New York, N. Y. (written discussion).—In connection with the use of silver-lead alloy anodes I should like to refer to our exhaustive tests carried out six years ago at Columbia University.[4]

U. C. TAINTON (written discussion).—The reduction in terminal voltage, as Professor Hanley has said, is important technically. I am keenly aware that the field has been barely touched upon in this paper. In addition to the extensive possibilities which remain in regard to other alloys, which we are still investigating, there is also the alternative method of securing lower voltage by the addition of the other elements to the solution. We have already obtained some interesting results in this field and hope to develop something commercially useful.

We have to disclaim the priority with which Professor Watts is good enough to credit us in regard to the general idea of lowering anode voltages by the use of other metals. As acknowledged in the paper, the first use of this scheme in metallurgy was by Dr. Fink and his associates in the application of the "Chilex" copper-silicon anode.

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[4] *Trans.* Amer. Electrochem Soc. (1924) **46**, 349; *ibid.* (1926) **49**, 85. Also U S. Patent.

As far as we know, we were the first to make use of the idea in zinc electrolysis. This opinion is confirmed by the patent offices of the world, which have nowhere brought forward a single citation against the grant of the patents.

O. C. **Ralston**, Clarkdale, Ariz. (written discussion).—This paper has been exceedingly interesting to me because two years ago I decided to take up the same project; namely, to work out an improved anode for an electrolytic zinc plant. While investigating the situation, I learned that Mr. Tainton had already had considerable success.

The authors stress the purity of their electrolytic zinc but have not discussed very much the point which was of greater interest to me; namely, the possibility of producing so-called chemical-grade manganese dioxide at the anodes for the benefit of the dry-cell manufacturers.

Chemical-grade manganese ore sells at $70 to $80 per ton for ore of 82 to 87 per cent. MnO_2 content. Some of the dry-cell manufacturers use chemical-grade ore on this type without any processing and others dissolve the manganese and prepare precipitated higher oxides of greater purity, so that the net cost of reasonably pure MnO_2 precipitate is usually over 5 cents per pound. Therefore, if MnO_2 can be precipitated on the anode with such an analysis and physical properties as to be acceptable to the dry-cell manufacturers, it would be worth almost as much per pound as is the zinc deposited simultaneously at the cathode.

The usual MnO_2 precipitate on the anodes of electrolytic zinc plants which operate with ordinary lead anodes contains from 5 to 20 per cent. of lead, probably as lead peroxide. This lead makes the MnO_2 precipitate unsuitable for battery manufacture, for the reason that it seems to spoil the shelf life of the dry batteries. The low corrosion of Mr. Tainton's alloy anodes suggests that his precipitate has either already met the specification of battery manufacturers or is approaching it. If not already satisfactory, apparently a joint research with one of the larger dry-cell manufacturers would be justified.

I am informed that the premium on the specially pure zinc manufactured at Kellogg is about $25 per ton, or 1.25 c. per lb. zinc. One can look for only 1 lb. MnO_2 for every 10 lb. of electrolytic zinc under present known methods of operating the electrolytic zinc cell. Anode efficiency in precipitating MnO_2 is low. However, this would correspond to a further increase of about 10 per cent. in the economic output of the electrolytic zinc cell and is well worth consideration. The special introduction of manganese into electrolytic zinc plant solutions for the purpose of heavier production of MnO_2 introduces certain problems in operation which are not insurmountable. It would mean a more rapid generation of sulfuric acid during the depletion of the electrolyte in zinc, but for such a contingency as this the Tainton process of electrolysis is well adapted.

At the time of my investigation of the market for chemical-grade manganese ore, it was sufficiently large to justify at least one large electrolytic zinc plant going into the business on the ratio of 1 lb. MnO_2 for every 10 lb. zinc produced.

The alloyed anodes described in this paper are definitely a marked step in advance of electrolytic zinc practice and I have tried to point out that full advantage of these anodes has not yet been taken.

High-zinc Slags in Australia

By Philip S. Morse, Brookline, Mass.

(New York Meeting, February, 1929)

The Australian lead-smelting plants began to use charges carrying high zinc percentages somewhat earlier than was common with American plants. When lead smelting first started in Australia the immense deposits of oxidized lead at Broken Hill were available but later these oxidized ores became exhausted and were replaced by sulfides. This was before the days of flotation and these sulfides when concentrated carried about 1 per cent. of zinc to 6 per cent. of lead.

The works of the Sulphide Corpn. at Cockle Creek was perhaps the pioneer in high-zinc slags but the Port Pirie plant of the Proprietary Co. was a close and very able second and has continued the work to the present time. The Cockle Creek lead furnaces remained in operation until shortly after the war when most of the Broken Hill companies combined and took over the Port Pirie plant to treat their concentrates.

As it was the practice both at Cockle Creek and Port Pirie to carry 20 per cent. or more of lead on the charge, it was found that under these conditions of large amounts of zinc in the charge, good roasting was absolutely necessary and matte production was unwise. The Huntington-Heberlein system of roasting was introduced and later the Dwight & Lloyd process. In this way, by roasting the sulfide portions of the charge as thoroughly as possible and making no matte, a method of smelting was developed where the slag over long periods carried as high as 20 per cent. ZnO without serious furnace troubles, and slags carrying even up to 33 per cent. ZnO have been reported.

The zinc slags were not very different in appearance from the ordinary slags produced in American practice except that they gave off more smoke and fume on tapping. The slag usually ran hot and freely, coming from the tap in a full stream. Normally there was little crust on the sides of the furnace and the tuyeres gave no trouble though they were always rather dark.

At the works of the Sulphide Corpn. it was the custom, on account of the labor situation, to shut down the furnaces during the Christmas holidays, thus giving a campaign of about 12 months. The furnaces at the end of the run usually showed no crust on the sides, though crusts always occurred at the back and front ends. A furnace campaign of 22 to 23 months has been made at this plant where both sides of the shaft were left totally free from crust. In this particular case the usual skin of fume on the bricks was not present, but on the contrary, the firebrick lining was found to be badly worn from the continual attrition of the charge. In these zinc campaigns the bullion in the crucibles usually remained quite hot, smoking when tapped. The syphons, however, often became choked with copper dross and required constant attention. The bullion contained a large amount of dross in which was practically all the copper on the charge. This dross was removed from the lead in a softening furnace and later made into a low-grade copper matte and then concentrated.

In calculating these high-zinc slags it was the practice to cut the lime and silica as the zinc increased. Iron was from the first recognized as a carrier of the zinc. A typical slag made at the Cockle Creek works (February, 1903) was as follows:

SiO_2, Per Cent.	CaO, Per Cent.	FeO and MnO, Per Cent.	ZnO, Per Cent.	Al_2O_3 Per Cent.	MgO, Per Cent.	S, Per Cent.	Pb, Per Cent.
22.0	13.0	33.5	14.5	8.0[a]	4.0[a]	3.0[a]	1.2

[a] Approximate.

The lead content of this particular slag is unusually low, the average being over 1.5 per cent. (wet determination).

The slags at the Port Pirie works commonly carried less iron than those at Cockle Creek. Typical slags at the Port Pirie works are given below:

	(1) 1918*	(2) 1922†	(3) 1922‡	(4) 1927§
SiO_2, per cent.	23.4	20.7	18.4	22.3
FeO, per cent.	34.4	28.0	20.3	29.6
MnO, per cent.		4.5	5.1	
CaO, per cent.	12.2	11.6	8.9	14.4
MgO, per cent.	Trace	2.0		
Al_2O_3, per cent.	5.0	6.2		6.3
ZnO, per cent.	20.1	20.9	33.3	21.8
S, per cent.	2.5	2.7		2.1
Cu, per cent.		0.19		
As, per cent.		0.02		
Sb, per cent.		0.05		
Cl, per cent.		0.077		
Pb, per cent.	1.5			

* Private communication from Gilbert Rigg (1918).

† *Proc.* Australian Inst. Min & Met. Engr., No. 47, 173. Slag stated to be an average over half-yearly period.

‡ *Ibid.*, 173.

§ Outline of Mining and Metallurgical Practice in Australia. Paper read at Second Empire Mining & Metallurgical Congress, Montreal (August, 1927) 44.

Australian zinc slags have been always marked by high sulfur and lead percentages. While low lead determinations were at times reported, a normal high-zinc slag at the Sulphide Corpn. Works carried 1½ per cent. or more lead, as determined by wet methods, and the Port Pirie slags were believed to carry about the same amount. The sulfur contents of slag at the Sulphide Corpn. Works were from 2 per cent. upward, and almost invariably as the sulfur contents increased, the lead percentage increased, though more slowly. It was found that slags carrying as high as 3½ per cent. sulfur or even 4 per cent. ran well and did not crust the furnace appreciably, but when the percentages of sulfur rose to more than this amount the presence of the familiar zinc mush, which most lead smelters have known to their sorrow, made itself felt and furnace troubles began. This zinc mush, from analyses made of the small particles that floated on the slag pots, gave analyses of 8 to 15 per cent. lead and about the same range for zinc; sometimes the zinc was higher than the lead and sometimes lower. The balance of the mush seemed to be iron and sulfur. Port Pirie has reported a mush or surface matte carrying Pb, 7.8; Zn, 8.6; FeO, 45.4, and S, 15.2 per cent., with small quantities of $SiO_2.CaO$ and other slag-forming materials.

Some of the characteristics of this surface matte would indicate that perhaps it was formed after tapping but the sulfide of zinc was present in the furnace and it is this that caused difficulties. Any oxide of zinc seemed to be easily taken care of in the slag. The Huntington-Heberlein process produced in the roasting a good deal of sulfate and the fumes from the top of the furnace showed that there was a good reaction between lead sulfates and lead sulfides above the smelting zone. In fact, it was often the custom to add small amounts of any lead sulfide that was free from zinc, unroasted, to the charge, with satisfactory results and of course without producing any matte.

The question of the action of zinc in the furnace and of its combination in the slag has always been an important one to all lead smelters. So far as the writer knows, no definite proof has been adduced as to how it actually is combined in the slag. It would seem reasonable to suppose, however, that most of the unoxidized zinc in the charge, if no matte is produced, passes through as a zinc sulfide or subsulfide and dissolves in the slag. The oxidized portion of the zinc probably takes the form of a spinel. Spinels are well-known constituents of slags and have been recognized in blast-furnace work. $ZnO.Fe_2O_3$ forms readily even at the temperature of roasting furnaces. In lead blast-furnace work, especially in those furnaces where high pressures are used, some of the more reduced forms of zinc-iron spinels which are known to exist may be formed. If much alumina is present in the charge, one would expect the formation first of a zinc-alumina spinel, the balance of the zinc forming a zinc-iron spinel. These reactions would of course be modified by large quantities of other elements on the charge, as for example, MgO which, at one of the American Smelting & Refining Co. plants, has been found to combine when large quantities are present, either in the furnace or after tapping, to form a magnesia-alumina spinel.

If we try to apply these theories of slag combination to the types of Port Pirie slags given here we should have to assume a more acid iron-lime silicate than is usually considered advisable, but if on slag 1 we take a bisilicate ratio of about SiO_2, 100; FeO, 52; CaO, 52, we should get a combination something as follows:

	Percentage Slag Combination	SiO_2, Per Cent.	FeO, Per Cent.	CaO, Per Cent.	ZnO, Per Cent.	Al_2O_3, Per Cent.
Silicate	47.8	23.4	12.2	12.2		
$ZnO.Al_2O_3$	9.0				4.0	5.0
$ZnO.Fe_2O_3$	35.6		21.2		12.0	
	92.4	23.4	33.4	12.2	16.0	5.0

This would leave 1 per cent. of FeO, 4.1 per cent. ZnO and 1.5 per cent. Pb to combine with the 2.5 per cent. sulfur.

With all of the other types of Port Pirie slags, as published, the percentage of lead is not given, but slags 2 and 4 are subject to an analysis similar to the above-suggested combinations. For the very curious slag 3, carrying 33.3 per cent. ZnO, neither the sulfur nor the alumina is given though a note states that the sulfur in these slags is usually 3 per cent. With the data given it seems impossible to fit it into the theory suggested here unless we suppose a large percentage of S and that most of the zinc appears in the slag as the equivalent of a subsulfide.

DISCUSSION

R. C. Canby, Wallingford, Conn.—In regard to the alumina spinel, I had an experience when I was assistant superintendent at the Arkansas Valley smelter. The superintendent had gone away and the furnaces were in very bad condition on account of excessive zinc. I put on the charge, using the insoluble residue as the silica in an ore the insoluble of which was very high alumina. The slag looked like about a 35 per cent. silica. The analysis showed only 28 per cent. The furnaces were seen in good condition. It makes me wonder whether this reference of Morse's to the alumina spinel may in a way account for that result.

Alumina slags seem to have good judgment. When alumina is needed as a base, it acts as a base, and when it is needed as an acid, it acts as an acid. Subsequently, under similar conditions I would put on a charge of alumina ore and very rapidly bring the furnace into good condition.

H. W. Hixon, New York, N. Y.—I was employed by the Teziutlan Copper Co. in Mexico several years ago. The ore ran about 3 or 4 per cent. copper and about twice that amount of zinc, all in the shape of sulfide.

When I went there, they used two kinds of coke, one imported from Pocahontas, the other produced in the state of Coahuila. These two cokes differed very largely in ash and also in physical properties. The Pocahontas coke is light and free from ash; the Coahuila coke is produced in by-product ovens and is in very large pieces. Frequently we would get a piece weighing 12 or 15 lb. This ore being all sulfide, when smelted in a blast furnace caused much trouble. The two kinds of coke behaved very differently in the furnace. The big, coarse coke made no trouble, but the fine, high-carbon coke made the furnace run badly. The oxidation of the sulfides in the furnace was affected by the coke. I abandoned the use of the Pocahontas and depended entirely on the Coahuila coke.

Also, under certain conditions the tuyeres would remain bright and the settler of the furnace would be free of mush, and it would be much easier to tap the settler. Actually I tried to reproduce those conditions of free running, regardless of what one might think about the necessities of the case, and I found that running on a slag that was about 42 per cent. silicate, about 22 per cent. CaO and 15 or 18 per cent. FeO, I could get along without much difficulty and smelt more tons of ore and fewer tons of charge, because we carried a higher percentage of ore on the charge in doing that.

The trap in the spout from the furnace to the settler had to be removed and replaced by a clay dam because zinc mush would accumulate in the tap hole and close it, causing slag and matte to run out of the tuyeres and frequently causing a freeze up. When the clay dam cut down enough to allow the blast to blow out of the tap hole this would blow out the zinc mush and enlarge the tap hole. After a short time the blast would be shut off and with a supply of clay ready a new dam would be put in the spout and the blast turned on again. Without this practice of the clay dam in place of a permanent trap the furnace would freeze up, no matter who was running it.

We had a power line from the smelter to the power house, which was in a canyon about 6 miles away. One day this power line was struck by lightning and everything went down. It was customary to wait for a considerable time before tapping out the furnace to see whether it would be possible to put in a fuse or put back the starting devices and go ahead. In the meantime, the furnace would be standing, with the uncertainty of whether the tap hole would remain open or would close. The Mexican foreman was on top of the settler punching a rod in the tap hole. I was a bit nervous about whether the thing was going to freeze up or whether we had better tap the furnace out on to the floor. There had been some wood burning on top of the crust of the settler to keep the hole open where the slag overflowed into the granulator. I stepped on the place where the fire had been, and went through. I went down into that settler up to my knee and was immediately blown out. The gas generated by my shoe and my trousers blew me out of the settler. A man who was working near by sprinkled oil on my shoe and I walked away and got well, and I have a perfectly good leg in spite of having gone down in the settler.

G. C. Stone, New York, N. Y.—I was in Port Pirie in '18 when they first successfully ran the lead furnaces on slags high in zinc. The methods were based on experiments made by Gilbert Rigg, one of our members. The first thing he did was to bury a pot of liquid slag in a large pile of sand so as to cool it very slowly; he then made thin sections of the slag starting at the outside of the pot and working in to the center, and examined them under the microscope. The outer layers did not show much but as they neared the center, where they had cooled more slowly, they showed three and sometimes four minerals very plainly. There were black octahedra which were either magnetite or franklinite, or both, purple octahedra of gahnite, zinc spinel ($ZnO.Al_2O_3$), and a groundmass of an orthosilicate, one of the olivine group which showed up plainly with polarized light. In some cases a fourth mineral, which had not been identified, could be seen in very small quantity.

Silicate of zinc has a relatively high melting point and a considerable difference in temperature between its softening and flowing point; it does not dissolve readily in the other silicates and therefore gives a mushy and unsatisfactory slag. If, on the other hand, it is present as a spinel $Fe_2O_3.ZnO$ or $Al_2O_3.ZnO$ it dissolves perfectly in the silicates and does not give trouble.

Rigg based his charges on the necessity of having all of the zinc as a spinel (gahnite or franklinite) and all of the silica as an orthosilicate of CaO or MgO (SiO_2, 2RO); that is, O in ZnO = ⅓O in Fe_2O_3 and Al_2O_3 and O in SiO_2 = O in CaO and MgO and MnO, if present. With this charge he obtained uniformly excellent results, making slags containing as much as 24 per cent ZnO that worked well in every way, being low in lead and permitting a larger output from the furnace than ever before. Incidentally, it required less iron than the slags previously made, which reduced the cost, as iron was the most expensive flux used.

In lead smelting the temperature is comparatively low and the reducing conditions are weak, so that all of the bases but PbO go into the slag. I was interested to see whether the slags made at higher temperatures and with stronger reducing conditions had similar compositions. Fortunately our own work gave me records to determine this.

In making oxide of zinc the temperatures are higher and the reducing conditions much stronger than in lead smelting, but the composition of the residuum (slag) is similar. A franklinite residuum shows under the microscope the same structure of magnetite crystals in a groundmass of an orthosilicate. Ignoring the iron, which always forms Fe_3O_4 and takes care of itself, if O in SiO_2 + ⅓O in Al_2O_3 = O in CaO, MgO and MnO the ZnO in the residuum will be at a minimum with any given burden and kind of ore.

In making spiegel and ferromanganese the temperatures and reducing conditions are higher than in any other blast-furnace smelting. Here the manganese lost depends largely on the burden and rate of driving, being greater with high burdens and fast driving. For any given burden and rate of driving it is lowest when the O in SiO_2 + ⅓O in Al_2O_3 = O in CaO and MgO. If there is not enough CaO and MgO in the charge to satisfy this equation the slag will tend to take up enough MnO to neutralize the SiO_2 and Al_2O_3. This can be partly prevented by a light burden and slow driving which, however, are not economical.

Some years ago, when our ore was running unusually low in Mn and high in Fe and we had contracts for high-grade spiegel, we kept the Mn in the cinder down to 2 per cent. for several months. Ordinarily it is more economical to run faster with higher burdens and lose more Mn.

In all cases it is necessary to base the composition of the slag on the O ratio and not on the percentages of the ingredients.

G. C. RIDDELL, New York, N. Y. (written discussion).—Philip Morse's paper is an interesting résumé of certain facts as to high-zinc slags. It would be still more interesting if the writer were to go a bit further into the subject and reflect the operating conditions—strenuous abnormalities of smelting procedure—that permitted these (at the time) extraordinary zinc slag burdens. For it is within bounds to say that up to 1920 or thereabouts the Australian smelters were nonchalantly swinging along with a slag-zinc 50 to 70 per cent. higher than the limits then regarded as feasible at American lead furnaces.

In more than one instance Australian metallurgy has distinctly blazed the trail for the rest of the world. In the early phases of two arts, flotation and zincy-lead smelting, Australians have led the way.

In 1916, I was called to Australia from Montana—where American lead smelting had been in the front trench as regards slags high in zinc—as consultant in the modernization of the Port Pirie smelter. Literally, I had to rub my eyes on arrival. Speaking generally, I had never seen in the United States of America anything to approach the smoothness and ease with which these two Australian plants (Port Pirie and Cockle Creek) were handling high zinc.

Each of these plants had a unique way of doing the trick. There had been almost no exchange of views between them; their methods, though far apart, were both effective. One plant—Cockle Creek, where Mr. Morse and the Evans brothers, Fred and H. A., had been for some years in charge—was in better shape than the other, as to method and general economy. This Cockle Creek plant was shut down shortly after the war and the particular point of its unique smelting procedure has not, to my knowledge, been recorded at large; at the time, in fact, its advanced technique was hardly recognized by the plant itself, as it had grown to its effectiveness in gradual steps, and with little reference to the rest of the world. As it remains practically without appreciation by the smelting fraternity, it is perhaps fitting that one or two of the high points be emphasized.

The following extracts are from my notes and letters of the period. I discuss the older plant first.

Port Pirie (under date of July, 1917).—"It was found in July, 1916, that a notably smooth furnace operation, on slags carrying 12 to 16 per cent. ZnO, was in progress at Pirie, involving some unusual methods of procedure:

"1. High return of slag, 65 to 75 per cent.
2. High slag-to-bullion ratio, 3.2 to 1.
3. High fuel, 16.5 to 17.5 per cent.
4. High blast, 56 oz.
5. Extremely low SiO_2 in slag, 20 per cent.
6. Dynamite bombing.
7. Tapping lead and slag at both ends of furnace.

"The three first points have undergone radical changes during the last 12 months, and can now be referred to as costly excesses which have been corrected. The four last points have much to do with the comparative immunity from furnace trouble that has long existed at Pirie on these high-zinc slags, and these features will doubtless remain of importance in future practice.

"For some years past, Pirie slags have averaged: 21 to 22 per cent. SiO_2, 6 per cent. Al_2O_3, 35 per cent FeO, 15 per cent. CaO, 1 per cent. MgO, 12 to 15 per cent. ZnO, 2.5 per cent. Pb, 1 oz. Ag, 3 to 4 per cent. sulfur. Pirie furnaces have, until the last year, carried 16.5 per cent. coke on the ore charge (ex slag and coke), and a 65 per cent. slag return. Blast furnaces are 206 by 56 in. at tuyeres. An 18-ft. ore column is carried with blast pressure of 56 oz.; smelting speed was slightly over 400 tons total charge per furnace day, up to a short time ago. Of this 400 tons, about 40 per cent. was foul slag returned from the dump, carrying 21 per cent. SiO_2, 38 per cent. FeO, 15 per cent. CaO, 2.5 per cent. Pb, and 11 per cent. Zn; so that the rate of actual ore-smelting was rather low—some 300 tons per 100 sq. ft. of hearth area. Percentage of Pb on ore charge ranged from 33 to 39.5; percentage of roasted product, 70 to 84.

"With the object of keeping the furnace "open"—*i. e.*, washing out crusts and keeping slag and lead taps working freely in presence of high zinc—some 600 tons of old dump slag were quarried and smelted daily, in former operations. This foul slag carried 1 to 3 per cent. less ZnO than the slag made at furnaces and served to appreciably relieve the zinc situation. This excessive slag return had been considered a necessity here for years, it having been the belief that a slag-to-bullion ratio of about 2.75 to 1 was essential."

It was demonstrated at Pirie in 1917, after reorganization of the feeding system, lowering of the smelting zone, introduction of type slags, closer coordination of roasting and smelting operations, and a general tightening up of furnace conditions, that the amounts of coke and return slag could be dropped to standard American practice, 15 to 20 per cent. and 12 to 12.5 per cent., respectively. This was done, with the zinc burden steadily increasing and slag leads inclined to improve.

For years, dynamite bombs had played an important part in Pirie furnacing. They were, in fact, the very backbone of this high-zinc work; without this muscular metallurgy it is probable that Pirie furnaces would have succumbed to crucible accretions long before the high zinc levels were attained. Used sparingly at a few other plants, and absolutely condemned at most, these dynamite pills, taken once an hour for years past, without a single serious accident, are perhaps the most striking novelty of a unique scheme of procedure which successfully handled at Pirie these high-zinc charges.

Type Slags.—Up to 1916, Pirie had been innocent of any sustained use of type slags. My notes carry the following comment (1917): "It has been interesting to note the changed behavior of furnaces since the adoption of definite type slags. The actual value of type slags may be a disputed point among lead metallurgists; adherents of the theory claiming cleaner slags, fewer accretions, and smoother general operation of furnace, while skeptics pronounce the connection between type slags and these advantages entirely imaginary. But it is to be noted that every one of the advantages claimed for type slags has come through at Pirie since their adoption. Whether the same improvement would have been accomplished, under the circumstances, had slags been run without regard to type, remains an open question.

"If zinc drops, in future operations, the tendency for SiO_2 in Pirie slag to increase toward the 28 to 32 per cent. normally used at other points will probably result in faster smelting. With present heavy amounts of Al_2O_3 and ZnO in the slag, it is probable that one, or both, of these elements are forced into an acid role by the scarcity of SiO_2, and any appreciable increase of the latter would upset the present slag equilibrium. Slag SiO_2 is now several per cent. higher than was carried up to a year ago, but a further change toward normal lead slag types, as far as SiO_2 is concerned, seems unwise until zinc drops. Previous to July, 1916, when fused analyses were established, SiO_2 determinations at Pirie had been several per cent. in error, on low side, due to failure to decompose slag samples in acids.

"A notable feature of Port Pirie metallurgy is the low SiO_2 carried in the slag. Some 3 per cent. Al_2O_3 is associated with this SiO_2 as an acid element, but these two acids together form a radical that is still very low, and the smooth handling of zinc is directly connected with this peculiar slag structure. The comparative absence of matte in these Australian operations has long been understood in metallurgical circles abroad to be involved with, or caused by, this low SiO_2 content of the slag. That this connection exists, I now feel sure, and the explanation seems to lie in the marked absorptive power of a low-silica slag for zinc oxide. When the silica is carried higher in the slag, and the zinc is less readily absorbed, a matte forms, containing the zinc that is thrown out of solution from the slag.

"Another explanation of this low matte fall with low slag silica may possibly be found in the presence of calcium sulfide in these slags. On a low-silica slag type some of the lime may go into combination with sulfur; I have found as much as 6 per cent. S in Pirie slags.

"Another cause of abnormally low matte fall at Pirie and Cockle Creek is the pyritic action which exists in the furnace shaft under the high blast pressures used—50 to 70 oz. Pirie chilled slag samples refuse to decompose in acids without fusion, indicating pyritic action or peroxidation of Fe."

Matte at Port Pirie.—There is a notion in many quarters that these Australian methods of smelting could be referred to as "making no matte." This Utopian situation was certainly *not* the condition at Pirie, though a comfortable impression prevailed at the plant itself that local matte fall, if any, was negligible. On this, my notes have the following:

"Absolutely no matte is brought back into the furnace charge; whatever matte comes out of the furnace tap-hole is (unless the testing ladle that is thrust down into the butt of each slag pot draws up a quantity of bullion) removed directly to the dump where the whole pot (shell, matte, and molten slag) is thrown over. If matte settles out in the half-mile trip to the dump, it goes over with the rest. There is every indication of a considerable metal loss on dump which can be saved in large part when proper settling facilities are provided, as a matte fall of 3 to 4 per cent. is usually existent. Samples of top matte (zinc mush) here, analyze 18 per cent. Zn, 10 per cent. Pb, 11 per cent. S, 13 per cent. SiO_2, 25 per cent. Fe."

Cockle Creek.—A different order of things prevailed at this smaller lead smelter in New South Wales. Over a period of years, at this plant of the Sulphide Corpn., a system of zincy-lead metallurgy had been developed that was far and away ahead of that of any plant in the world at that time. Cockle Creek, in a word, was putting a zinc burden through lead furnaces with a technique that was superb. High tonnage, good slag leads, and easy furnace conditions attended this operation.

Slag zincs ranged at about the same levels as Pirie, and a small matte trap was in use at the furnaces. On a 60 by 138-in. furnace with 60 to 70 oz. blast the following conditions were observed, described by Mr. Evans as an ordinary day's run (but probably one of the best performances of which the furnace was in those days capable):

Charge (ore and flux), short tons (587 tons per 100 sq. ft. hearth), 337.5.
Returned slag shells, per cent. 17.8.
Coke, per cent., 11.1.
Ratio of slag to bullion, about 2 to 1.
Per cent. roasted product (H.-H.) on charge, 73.3.
Per cent. Pb on roasted product, 49.0.
Per cent. S in roasted product, 4.0 (2 per cent. sulfates).
Analysis slag: SiO_2, 23.6; FeO, 35.5; MnO, 3.2; CaO, 15.0; ZnO, 13.1; Pb, 0.6.
Matte fall undetermined, but low.

On analyzing the Cockle Creek performance, I found one predominant condition maintained. My files carry the following memo: "Significance of the high-column smelter in use by the Sulphide Corpn. at Cockle Creek: Examining rather carefully into this matter during my visit at this plant, I find that this high-column business is of importance; we little realized how close to the mark we were shooting when we specified an increase in the height of ore column at the Pirie furnaces.

"That Cockle Creek is getting and maintaining slag Pb's well under 1.5 per cent., smelting at a 550-ton clip, and with 11.0 to 11.5 per cent. coke on charge, is a certainty. The prime factor in this is high column and high pressure, 28 to 30 ft. of ore above the tuyeres, 60 to 70 oz. blast. A high pressure is of no benefit without the high column; one would simply burn up excess coke; the two together, however, are a veritable panacea for zinc troubles at Cockle Creek.

" 'Speed, drive, hurry-up,' is the word at Cockle Creek. There is abundant evidence here that they were not able to maintain good slags (0.6 to 1.5 per cent.) and clean free crucibles, as now, until this high furnace went in. As soon as the high column became available their speeds went up, and crucible crusts and dynamite bombing disappeared. Whenever blast pressure nowadays makes an appreciable drop, due to coarse run of charge, etc., reduction goes off.

"On the low furnace they were barring accretions every three weeks; the advent of high column marked the end of wall accretions, for the most part. Three to four months is now the campaign. The charge itself is believed to catch the accretions; *i.e.*, the descent of fines at sides washes down the shaft walls. The fumes ascending at the center of the furnace through the coarser charge seem to deposit their accretions down to the smelting zone.

"This Cockle Creek procedure is a scheme of things that, under the conditions, is in advance of smelting performance in the States. It was something of a revelation to see the comparative ease and smoothness with which Pirie furnaces, even under the old conditions, were handling slags carrying 13 to 14 per cent. ZnO. Just as this Pirie slag manipulation is ahead of similar work in America, so is Cockle Creek a good length in advance of Pirie, in this matter of clean, fast work on high-zinc charges. The outstanding point here is 'Drive the furnace.'"

F. M. Smith, Spokane, Wash. (written discussion).—It must be admitted that Mr. Morse's theory is very ingenious, but I am not prepared to pass judgment as to the correctness of the theory, nor am I able to expound a theory of my own as to just how zinc does enter into slag compositions. I am not favorably impressed with the practicability of the theory, because of the fact that for very high zinc contents the strict application of the theory requires such low-silica, high-iron compositions. At most lead smelters in the United States iron is an expensive flux, and this has always been particularly true at East Helena; hence to strictly follow the spinel theory, the amount of iron required would be prohibitive from an economical standpoint. Hence, the Morse spinel practice was never adopted as a permanent proposition at East Helena.

Mr. Morse's spinel theory apparently was not adopted at the Port Pirie Smelting Works, South Australia, as is shown by the following analysis of the slag which Mr. Robertson of that works had been running for a long time, as late as 1919, with good results:

	SiO_2, Per Cent.	FeO, Per Cent.	MnO, Per Cent.	CaO, Per Cent.	MgO, Per Cent.	Al_2O_3, Per Cent.	ZnO, Per Cent.	S, Per Cent.	Cu, Per Cent.	Pb Per Cent.	Ag, Oz.
Robertson slag.....	24.0	30.7	5.2	13.6	1.2	4.2	16.7	2.8	0.3	1.5	0.4
Morse slag	18.3	41.1	2.0	10.4	1.6	4.0	16.7	3.0	0.2	1.0	0.4

16.7 per cent. ZnO is 13.4 per cent. Zn

Through the kindness of J. D. MacKenzie, manager at East Helena, I am able to give the following analyses of recent East Helena slags:

Cu, Per Cent.	Pb, Per Cent.	FeO, Per Cent.	SiO_2, Per Cent.	CaO, Per Cent.	MgO, Per Cent.	ZnO, Per Cent.	Al_2O_3, Per Cent.	S, Per Cent.	MnO, Per Cent
0.3	0.86	35.0	23.8	13.8	1.0	17.8	3.5	3.1	1.0
		34.9	22.2	12.8	1.0	20.4	3.8	3.2	1.0
	0.90	36.0	31.4	12.1	1.0	21.7	3.7	3.3	1.0

These slags indicate a higher silica and a lower iron than the Morse theory would require. They show remarkably good reduction; obtained, however, at a heavy sacrifice of speed or tonnage. Apparently good reduction is stressed rather than tonnage at East Helena.

Two recent Herculaneum slags and three Trail slags are shown on p. 213. Those from Herculaneum show high zinc with relatively high silica and fair reduction.

Trail, as everyone knows, has a big surplus of iron in the concentrates from the Sullivan mill. On the other hand, it is hard up for silica; hence, as iron costs nothing there, and silica is really an expensive flux, it is natural that the Trail slag should run lower in silica and higher in iron than at the average lead smelter in the United States. These Trail slags show rather poor reduction, but I understand that this

Herculaneum

Date	Cu	Pb	FeO	SiO_2	CaO	MgO	ZnO	Al_2O_3	S	MnO
Mar., 1928......		1.5	34.0	25.0	8.5	4.5	18.75	3.0	3.0	
Mar., 1928......		1.5	37.0	22.0	9.0	5.0	21.50	3.0	3.0	
Trail										
May, 1928......		1.9	37.3	20.6	12.0	3.2	18.40	3.2		0.6
Aug., 1928.......		2.2	29.9	19.8	10.0	1.3	19.10	3.8	2.2	
Sept., 1928......		1.8	42.4	18.9	10.5		20.70			

does not worry our Trail friends because they propose to retreat this slag to recover the zinc and, incidentally, the lead. In other words, Trail appears to lean toward speed and large tonnage rather than to low-lead slags.

At the Bunker Hill smelter we were running a slag like this in April, 1928, which was before we began to smelt the zinc residues from the Sullivan electrolytic zinc plant:

FeO	MnO	SiO_2	Al_2O_3	CaO	MgO	S	ZnO	Cu	Pb
34.5	2.8	24.8	6.6	16.8	1.6	1.6	9.7	0.10	1.4

A. F. Beasley reports that this was an exceptionally good slag, in that the furnace ran very fast and gave us a large tonnage. About October, 1928, we began to receive the zinc-plant residues, and our troubles began. The furnaces slowed up and we had great difficulty in holding reduction. The following gives the complete analysis of a composite slag sample for December, 1928:

Pb	FeO	MnO	SiO_2	Al_2O_3	CaO	MgO	S	ZnO	Cu	Ag
1.6	35.6	2.7	23.0	5.1	14.1	2.0	1.8	13.3	0.10	0.10

One more recent slag, that of Feb. 1, 1929, was as follows:

Pb	FeO	MnO	SiO_2	Al_2O_3	CaO	MgO	S	ZnO	Cu	Ag
1.2	33.5	2.4	24.2	5.5	14.4	2.0	1.8	14.5	0.05	0.08

I think all lead metallurgists are agreed that high-zinc slags require relatively low silica and high iron contents. I do not believe that there is any fixed rule for determining the exact percentages of SiO_2 and FeO for varying ZnO contents. This is something that each metallurgist has to work out for himself according to the peculiar conditions at his own plant. If his cost of iron is low, he can afford to run high iron in his slag; on the other hand, if he has good smelting margins on siliceous ores, he will strive to run as high silica in his slag as conditions will permit. Some slags run well and others do not, and I believe that it is simply a question of "trial and error," and one that each man must work out for himself.

R. P. RAYNOLDS, Denver, Colo.—About 1907 I had some experience at Durango with a high-alumina slag which Mr. Morse's interesting paper calls to mind. Unfortunately, many of my notes have been lost in moving, so that I am unable to give much detail, but as I recall it, the slag varied from 6 to 10 per cent. Al_2O_3. A discrepancy between the charge as figured and the slag analysis caused us to investigate, and we found that the silica determination as ordinarily made was between 3 and 4 per cent. high. After treatment with hydrofluoric acid there was a residue left. This error in the determination accounts for the discrepancy between charge calculations and the slag analysis. I recall that we made a great many determinations but the only one that I can locate showed the following slag analysis:

Insoluble	Fe	Mn	CaO	Zn	Al_2O_3	Pb	Cu
29.4	28.7	0.07	13.4	8.0	7.3	0.9	0.41

The insoluble was fused with alkali, taken up with acid, dehydrated and the silica expelled with hydrofluoric acid, leaving a residue, which analyzed: Fe, 0.58; CaO, 0.07; Zn, 0.6; Al_2O_3, 1.28 per cent.

This insoluble residue was present whenever the alumina in the slag was over 6 per cent. and the higher the percentage of alumina, the larger was the proportion of the residue.

The various analyses of the residue checked closely enough to permit us to use a factor when a quick result was wanted without waiting for the final analysis of the residue. The analysis of this residue closely approximates a zinc-iron spinel and we were convinced that the presence of high alumina led to the formation of this spinel.

C. S. Witherell, New York, N. Y.—In many of the blast-furnace slags in which high iron is used to dissolve as much zinc as possible, if you figure the zinc as ferrozincate (FeO.ZnO) and deduct that from the composition of the slag under consideration, then figure the composition of the slag that remains, you will find a fairly well balanced, neutral slag in nearly every case. I have done that with the slag compositions reported used at the Trail plant and various other plants producing slags high in zinc. In fact, I think in those given in this paper you can figure the iron combined with zinc in the proportion of one FeO to one ZnO; that is, give to the zinc oxide its proportion of ferrous oxide, to form FeO.ZnO, deduct both from the slag, calculate the slag that remains, and for a majority of examples given you will find a fairly good type of slag, a slag that will act as a good solvent for dissolving the so-called ferrozincate, alumina, and other ingredients questionable as to role played in slag composition.

Formation of Insoluble Zinc Compounds during Roasting

By H. R. Hanley,* Charles Y. Clayton† and David Walsh,‡ Rolla, Mo.

(New York Meeting, February, 1929)

It is a well-known fact that the solubility of zinc compounds decreases when these compounds are roasted in contact with iron compounds, but descriptions of tests to quantitatively express the fact have not appeared in the literature. Possibly such an expression cannot be uniformly and exactly duplicated by different operators because of the variation in the personal equation. In a general way, however, the reactions between zinc compounds and iron compounds under roasting conditions will show a certain trend with changing conditions and the approximate quantitative expression of this trend appears reasonably well established by experimental work conducted for this purpose.

Ferrites and Marmatites

The name "ferrite" has been given to the compound of zinc and iron which has the formula $ZnO.Fe_2O_3$.

Among the questions pertinent to this subject are those involving inherent or natural ferrite, and also the synthetic or artificial variety. The inherent or natural product is formed when the mineral containing iron and zinc sulfides isomorphously crystallized is subjected to oxidation at roasting temperature. These isomorphously crystallized minerals are known, mineralogically, as marmatites, and contain ZnS and FeS in ratios of 2 to 1 up to 5 to 1 respectively. The zinc and iron sulfides form a definite mineral. They are firmly attached to each other and form common contact planes through which diffusion of the entities occurs during the roasting process. This diffusion of the molecularly intimate entities produces, during roasting, the compound referred to as ferrite, in proportion to the amount of iron originally present in the marmatite. For example, a marmatite from British Columbia containing 9.739 per cent. Fe and 56.87 per cent. Zn corresponds with the mineral having the formula 5ZnS.FeS. The ferrite produced in roasting, which has the formula $ZnO.Fe_2O_3$, contains 27.07 per cent. Zn and 46.40

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per cent. Fe. The ratio of zinc to iron in the ferrite is $\frac{27.07}{46.40}$ per cent. = 0.584. Therefore there is only 0.584 unit of zinc made insoluble per unit of iron present. Inasmuch as there are only 9.739 units of iron present in the marmatite, the total amount of zinc with which this iron can combine and thus render insoluble will be $9.739 \times 0.584 = 5.687$ units of zinc. If we refer these units of insoluble zinc to the total zinc present in the marmatite we see the percentage insolubility of the zinc (100 per cent. — soluble) will be $\frac{5.687}{56.87} = 10.00$ per cent. The treatment of the roasted marmatite with 4 per cent. H_2SO_4 solution, or ammonium chloride solution, will dissolve zinc to the extent of approximately 90 per cent. of the total zinc present.

A marmatite having a formula 3ZnS.FeS after roasting will produce a calcine having approximately 83 per cent. of the zinc soluble in weak acid or ammonium chloride. The preceding conditions are obtained regardless of the temperature at which the roasting is conducted, providing the marmatite is pure. This was verified by experiment. This definite solubility of the zinc naturally follows as a corollary because the iron oxide will combine only with a definite amount of zinc oxide in the ferrite compound, and since the iron present in the original mineral is in limited amount, the toxic effect of the latter will only be exerted in proportion to this limited amount isomorphously crystallized with the zinc as sulfide. In other words, the iron present in the marmatite mineral may be regarded as the potential iron in the ferrite.

TEST ON A SPECIAL MARMATITE

A marmatite assaying 48 per cent. Zn, 11 per cent. Fe, 30 per cent. S was roasted in an electric muffle at 625° C. for 6 hr. The sulfide sulfur present in the calcine was 0.2 per cent. The zinc solubility in the calcine was 88.6 per cent. This calcine was then baked for another 6 hr. at 900° C. The zinc solubility was 87.9 per cent.

Another portion of the same marmatite was roasted at 900° C. for 6 hr.; this showed 87.8 per cent. zinc solubility. These figures are in agreement with each other when the effect of the free iron is taken into account for the two roasts at different temperatures.

The characteristics of this marmatite are calculated as follows: 48 per cent. Zn as marmatite mineral, 8.23 per cent. Fe as marmatite mineral, 2.77 per cent. Fe as pyrrhotite; 11.00 per cent. Fe total.

When roasted at 625° C., 10 g. marmatite gave 10 g. calcine. The normal loss in weight was counterbalanced by a gain in weight due to the formation of $ZnSO_4$.

Combined Fe = 0.823 g. per 10 g. calcine.

Free Fe $= \frac{0.277}{1.100}$ g.

0.823 g. Fe × 0.584 = 0.481 g. Zn insoluble due to 0.823 g. combined Fe.

0.277 g. Fe × 0.584 = 0.161 g. Zn insoluble if Fe were 100 per cent. effective.

The actual insoluble Zn was: 10 g. calcine at 5.5 per cent. = 0.550 g.
Insoluble Zn due to combined Fe, as before = 0.481
Insoluble Zn due to 0.277 g. free Fe = 0.069

Segregation of Insoluble Zinc

The 0.823 g. Fe combined with 0.481 g. Zn at 100 per cent. effectiveness.
The 0.277 g. Fe is theoretically equivalent to 0.161 g. Zn.
The 0.277 g. Fe actually combined with only 0.069 g. Zn.

Effectiveness of free Fe $= \frac{0.069}{0.161} = 42.80$ per cent. (ferrite due to free Fe). Ratio of effectiveness between combined Fe and free Fe for the production of insoluble zinc compounds under this condition amounts to 42.80 per cent.

Effect of Free Iron

It was decided to ascertain the effect of free iron as pyrite, or pyrite plus pyrrhotite when present with marmatite in various ratios. Accordingly progressively increased weights of powdered pyrite were intimately mixed with powdered marmatite and the mixtures roasted for 8 hr. in an electric muffle pyrometrically controlled. There were two different temperatures selected for the roasting, which would approximate plant conditions; *e. g.*, 625° C. as might be used in the roasting division of an electrolytic zinc plant, and 900° C. as the maximum temperature that would be used in the roasting division of a retort zinc plant. The results of these roasting experiments are shown in Tables 1 and 2. Generalizing these results, there is shown that approximately 0.25 g. zinc is made insoluble per gram free iron when roasted 8 hr. at 625° C. At 900° C., however, the insoluble zinc formed is approximately 0.45 g. per gram of free iron. In either case the combined iron, as marmatite constituent, always renders zinc insoluble to the extent of 0.584 g. zinc per gram of combined iron. It follows, therefore, that the generalized effectiveness of the free iron as pyrite to form insoluble zinc is approximately 40 to 50 per cent. of the combined iron when the roasting temperature is maintained at 625° C. Similarly, free iron as pyrite has the general effectiveness of 80 to 90 per cent. of the combined iron when the roasting temperature is maintained at 900° C. The data obtained in the roasting experiment at 625° C. are shown in Table 1. Graphic interpretation of the essential data is shown as curves in Fig. 1. The similar data obtained at 900 ° C. are shown in Table 2; the graphic expression is shown in Fig. 2. These graphs show the following features:

1. Zinc solubility referred to free iron.
2. Ferrite formation; per cent. to total possible that could be formed if the iron were 100 per cent. effective.
3. Grams zinc made insoluble per gram of free iron. This can be interpreted in units if desired.

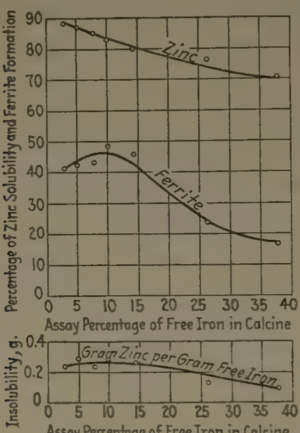

FIG 1.—MARMATITE WITH ADDITIONS OF PYRITE ROASTED AT 625° C.

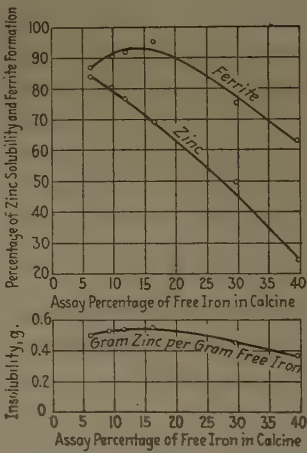

FIG. 2.—MARMATITE WITH ADDITIONS OF PYRITE ROASTED AT 900° C.

EXAMPLE OF THE METHOD USED IN CALCULATING DISTRIBUTION OF INSOLUBLE ZINC WITH COMBINED IRON AND FREE IRON

Calcine 9.—625° C. Roast 8 Hr.

	GRAMS
Combined Fe	0.823
Free Fe	0.577
Total Fe	1.400

0.823 g. Fe × 0.584 = 0.481 g. Zn insoluble due to 0.823 g. combined Fe
0.577 g. Fe × 0.584 = 0.337 g. Zn theoretically insoluble Zn if Fe were 100 per cent. effective. The actual insoluble zinc was:

(1.00 − 0.863) 4.8 g. Zn gives 0.658 g. insoluble zinc.

	GRAMS
Actual insoluble zinc	0.658
Insoluble zinc due to combined iron	0.481
Insoluble zinc due to 0.577 g. free iron	0.177

Segregation of Percentage Insoluble Zinc

The 0.823 g. Fe combined with 0.480 g. Zn at 100 per cent. effectiveness.
The 0.577 g. Fe is theoretically equivalent to 0.337 g. Zn.
The 0.577 g. Fe actually combined with only 0.173 g. Zn.

Effectiveness of free Fe $= \frac{0.177}{0.337} =$ 52.5 per cent. ferrite due to free Fe.

There is no quantitative relation between the variation of the zinc solubility and the percentage of ferrite formed,[1] for the reason that the altered ratios of zinc to free iron do not affect these features in the same

TABLE 1.—*Loss of Zinc Solubility of Marmatite* (100 *Per Cent. Insoluble*) *per Unit of Free Iron Present*

Calculated Solubility of Zinc if All (Free) Iron Were Absent = 90.00 Per Cent. Roast 8 Hr. at 625° C.

Sample Number	9	10	11	12	13	14	15
1. Marmatite, grams	10.00	10.00	10.00	10	6.66	5.00	10
2. Zn present, grams	4.8	4.8	4.8	4.8	3.2	2.4	4.8
3. Combined Fe present, calculated, grams	0.823	0.823	0.823	0.823	0.548	0.4115	0.823
4. Free Fe present, grams	0.277	0.277	0.277	0.277	0.182	0.1385	0.277
5. Fe (as pyrite) added, grams	0.300	0.600	0.900	1.500	2.720	4.100	None
6. Total free Fe present, grams	0.577	0.877	1.177	1.777	2.902	4.238	0.277
7. Total Fe present, grams	1.400	1.700	2.000	2.600	3.450	4.650	1.100
8. Free Fe in calcine, assay per cent.	5.06	7.62	10.00	14.23	26.20	37.70	2.77
9. NH_4Cl sol. Zn in calcine, assay per cent.	36.70	35.60	34.00	30.72	21.95	15.10	42.50
10. Total Zn in calcine, assay per cent.	42.20	41.70	41.00	38.40	28.70	21.30	48.00
11. Zn solubility in NH_4Cl sol.	87.00	85.40	83.00	80.00	76.50	70.90	88.60
12. Theoret. insol. Zn due to free Fe, grams. (Free Fe × 0.584)	0.337	0.512	0.688	1.037	1.693	2.475	0.161
13. Total insol. Zn, grams	0.624	0.702	0.816	0.960	0.720	0.659	0.548
14. Insol. Zn due to combined Fe, grams	0.481	0.481	0.481	0.481	0.320	0.240	0.481
15. Zn made insol. due to free Fe, grams	0.143	0.221	0.335	0.479	0.400	0.419	.067
16. $\frac{\text{Insol. Zn}}{\text{Free Fe}}$, grams	0.307	0.252	0.284	0.270	0.138	0.099	0.242
17. Ferrite[a] due to free Fe present, per cent.[b]	42.4	43.20	48.70	46.20	23.6	16.95	41.60

[a] In all tables, this phase is to be interpreted "per cent. effectiveness of free Fe for ferrite formation."

[b] These figures are obtained by dividing the figures of the 15th line by those in the 12th.

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[1] The phrase "ferrite formed" is to be interpreted "Effectiveness of free Fe for ferrite formation."

way. In general, with ores roasted at low temperature, the smaller the amount of free iron present, the greater is the opportunity for each particle of free iron to come in contact with particles of zinc, and since this contact is essential for ferrite formation, the condition that promotes it will yield the greatest percentage of ferrite. The percentage

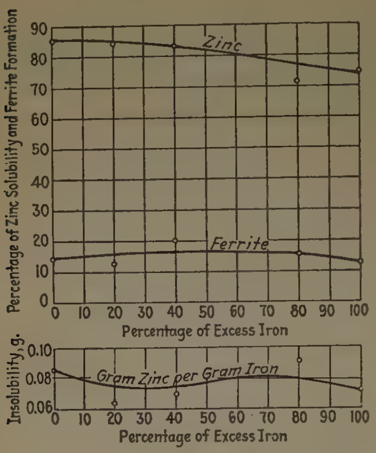

Fig. 3.—$ZnO + 2FeS_2$, increasing FeS_2 up to 100 per cent. excess of that required by formula Zn: 2 Fe. Roasted 8 hr. at 625° C. Rabbled every 10 minutes.

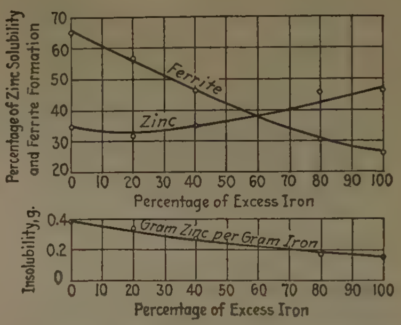

Fig. 4.—$ZnS + Fe_2O_3$ increasing Fe_2O_3 up to 100 per cent excess of that required by formula Zn: 2 Fe. Roasted 8 hr. at 625° C. Rabbled every 10 minutes.

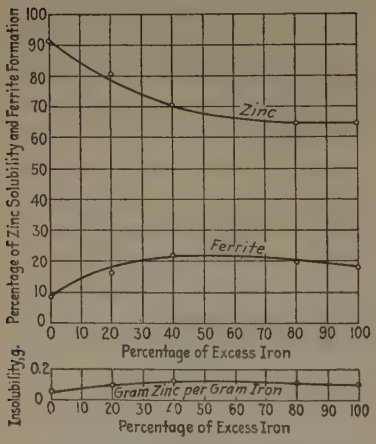

Fig 5.—$ZnS + 2FeS_2$. Increasing FeS_2 up to 100 per cent. excess of that required by formula Zn: 2 Fe. Roasted 8 hr. at 625° C. Rabbled every 10 minutes.

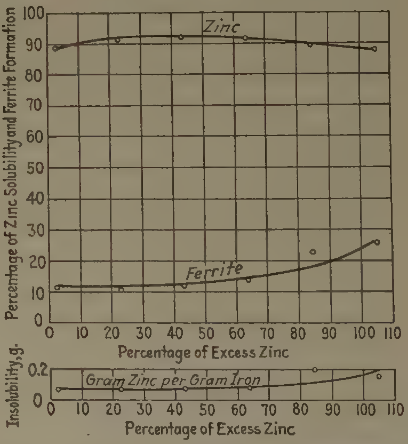

Fig. 6.—$ZnS + 2FeS_2$. Increasing ZnS up to 100 per cent. excess of that required by formula Zn: 2 Fe. Roasted 8 hr. at 600° C. Rabbled every 10 minutes.

of ferrite formed is the relation between the amount of insoluble zinc formed to the amount that could be formed theoretically in the compound $ZnO.Fe_2O_3$ due to free iron. The greater the amount of free iron present, the less opportunity there is for each particle of iron to come in contact with particles of zinc, for obvious reasons (*e. g.*, the particles of iron form aggregates and thus prevent contact of single particles of iron with particles of zinc). This does not hold for all ratios in the case of sulfides.

TABLE 2.—*Loss of Zinc Solubility of Marmatite* (100-*insoluble*) *per Unit of Total and per Unit of Free Iron Present*

Calculated Solubility of Zinc if All Uncombined Iron Were Absent = 90.00 Per Cent.
Roast 8 Hr. at 900° C.

Sample Number	16	17	18	19	20	21
1. Marmatite, grams	10.00	10.00	10.00	10.00	6.66	5.0
2. Zn present, grams	4.8	4.8	4.8	4.8	3.2	2.4
3. Combined Fe present, calculated, grams	0.823	0.823	0.823	0.823	0.548	0.4115
4. Free Fe present, grams	0.277	0.277	0.277	0.277	0.182	0.1385
5. Fe (as pyrite) added, grams	0.300	0.600	0.900	1.500	2.720	4.100
6. Total free Fe present, grams	0.577	0.877	1.177	1.777	2.902	4.2385
7. Total Fe present, grams	1.400	1.700	2.000	2.600	3.450	4.650
8. Free Fe in calcine, assay per cent	6.62	9.37	11.90	16.38	29.70	39.60
9. NH_4Cl sol. Zn in calcine, assay per cent	44.80	40.84	36.85	30.35	16.10	5.47
10. Total Zn in calcine, assay per cent	53.4	50.80	48.00	43.80	32.30	22.10
11. Zn solubility in NH_4Cl sol.	83.90	80.30	76.80	69.35	49.80	24.72
12. Theoret. insol. Zn due to free Fe, grams. (Free Fe × 0.584)	0.337	0.512	0.688	1.037	1.693	2.475
13. Total insol. Zn, grams	0.773	0.946	1.114	1.471	1.605	1.806
14. Insol. Zn due to combined Fe, grams	0.481	0.481	0.481	0.481	0.320	0.240
15. Zn made insol. due to free Fe, grams	0.292	0.465	0.633	0.990	1.285	1.56
16. $\frac{\text{Insol. Zn}}{\text{Free Fe}}$, grams	0.506	0.530	0.537	0.556	0.422	0.368
17. Ferrite due to free Fe present, per cent.[a]	86.7	90.8	92.0	95.4	75.4	63.1

[a] These figures are obtained by dividing the figures of line 15 by those of line 12.

The increase of free iron lowers the zinc solubility in marmatite ores, for two reasons: (1) it slightly increases the amount of insoluble zinc due to formation of ferrite; (2) it lowers the tenor of zinc in the calcine.

The following substances were mixed and roasted 8 hr. at 900° C., the rabbling being done every 10 minutes.

Series 1. Marmatite with increasing amount of FeS_2 at 625° C.
Series 2. Marmatite with increasing amount of FeS_2 at 900° C.
Series 3. ZnO and $2FeS_2$.
Series 4. ZnS and Fe_2O_3.
Series 5. ZnS and FeS_2.
Series 6. ZnS and $2FeS$ with increasing amount of Zn.
Series 7. ZnS and $2FeS_2$ with increasing amount of Zn 700° C.
Series 8. ABC $ZnS.FeS_2$ with great variation of FeS_2.

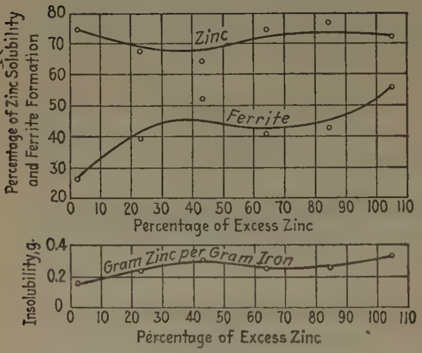

Fig. 7. $ZnS + 2FeS_2$. Increasing ZnS up to 100 per cent. excess of that required by formula $Zn : 2 Fe$.
Roasted 8 hr at 700° C. Rabbled every 10 minutes.

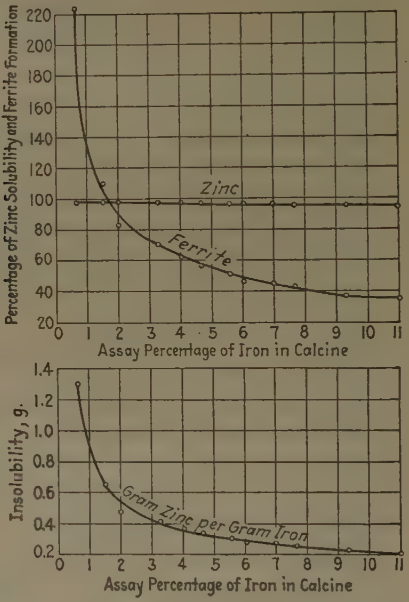

Fig. 8.—ZnS and FeS_2. Commencing with very small amount of FeS_2, progressively increasing.
Roasted 8 hr. at 625° C. Rabbled every 10 minutes.

Series 1, *Table* 1.—Marmatite plus FeS_2 roasted at 625° C. This series shows a gradual lowering in the zinc solubility with increase of FeS_2. The percentage of artificial ferrite formed, referred to uncombined iron, increases up to a certain point then falls. Apparently this is due to the lack of effectiveness of the iron when its ratio to the zinc is increased.

Series 2, *Table* 2.—Marmatite plus FeS_2 roasted at 900° C. The same general features are present in this case except that the artificial ferrite is much higher. From these data the effect of free iron on zinc solubility may be approximated for any tenor in zinc and iron. It is assumed that

TABLE 3.—*Results of Roasting* $ZnO + 2\ FeS_2$ *with Increasing Amount of* FeS_2 *Up to* 100 *Per Cent. Excess of That Required by Formula* $Zn:2\ Fe$

Roasted 8 Hr. at 625° C. Rabbled at 10-min. Intervals.
Reagents: ZnO Pure
FeS_2 = 44.4 Per Cent. Fe
4.07 g. ZnO = 3.27 g. Zn

Sample Number	6	7	8	9	10
ZnO, grams	4.07	4.07	4.07	4.07	4.07
FeS_2, grams	12.62	15.47	17.60	22.70	25.23
Fe present, grams	5.60	6.72	7.84	10.08	11.20
Excess of Fe, per cent	0.0	20.0	40.0	80.0	100.0
Total Zn, assay per cent	20.00	18.10	16.9	15.00	12.90
NH_4Cl sol. Zn, assay per cent	17.10	15.30	14.10	10.72	9.65
Zn solubility in NH_4Cl sol	85.50	84.60	83.20	71.50	74.80
Insol. Zn (actual), grams	0.474	0.503	0.549	0.932	0.824
Insol. Zn (theoretic), grams	3.270	3.930	4.578	5.880	6.490
Ferrite formed, per cent	14.50	12.80	20.00	15.84	12.70
Zn insol. per gram Fe, grams	0.0846	0.0634	0.0700	0.0924	0.0735

TABLE 4.—*Results of Roasting* $ZnS + Fe_2O_3$ *with Increasing Amount of* Fe_2O_3 *Up to* 100 *Per Cent. Excess of That Required by Ratio* $Zn:2\ Fe$

Roasted 8 Hr. at 625° C. Rabbled at 10-min. Intervals
Reagents: Blende Zn = 58.23 Per Cent.
Fe_2O_3 = 66.6 Per Cent. Fe
5.61 g. Blende = 3.27 g. Zn

Sample Number	11	12	13	14	15
Blende, grams	5.61	5.61	5.61	5.61	5.61
Fe_2O_3, grams	8.42	10.10	11.78	15.17	16.85
Fe present, grams	5.6	6.72	7.84	10.08	11.20
Excess Fe, per cent	0.0	20.0	40.0	80.0	100.0
Total Zn, assay per cent	23.90	20.80	18.90	15.35	14.95
NH_4Cl sol. Zn assay per cent	8.26	6.60	6.60	7.00	7.00
Zn solubility in NH_4Cl sol	34.55	31.70	34.90	45.80	46.80
Insol. Zn (actual), grams	2.140	2.230	2.120	1.770	1.730
Insol. Zn (theoretic), grams	3.270	3.930	4.578	5.80	6.54
Ferrite formed, per cent	65.4	56.80	46.3	30.5	26.4
Loss Zn solubility per unit Fe	1.59	1.59	1.43	1.13	1.04
Zn insoluble per gram Fe, grams	0.382	0.332	0.271	0.175	0.154

TABLE 5.—*Result of Roasting $ZnS + 2FeS_2$ with Increasing Amount of FeS_2 Up to* 100 *Per Cent. Excess of That Required by Ratio Zn*:2 *Fe*

Roasted 8 Hr. at 625° C. Rabbled Every 10 Min.
Reagents: Blende Zn = 58.23 Per Cent.
Pyrite Fe = 44.4 Per Cent.
5.61 g. Blende = 3.27 Zn

Sample Number	16	17	18	19	20
Blende, grams	5.61	5.61	5.61	5.61	5.61
FeS_2, grams	12.62	15.14	17.60	22.70	25.23
Fe, grams	5.60	6.72	7.84	10.08	11.20
Zn, grams	3.266	3.266	3.266	3.266	3.266
Excess Fe, per cent	0.0	20.0	40.0	80.0	100.0
Total Zn, assay per cent	20.00	18.00	17.20	14.40	13.00
NH_4Cl sol. Zn, assay per cent	18.20	15.00	10.60	9.35	8.36
Zn solubility in NH_4Cl sol	91.00	80.70	70.00	64.90	64.30
Insol. Zn found, grams	0.294	0.631	0.980	1.145	1.167
Insol. Zn (theoretic) (Fe × 0.584) grams	3.270	3.930	4.578	5.880	6.54
Ferrite formed, per cent	8.94	16.05	21.4	19.47	17.85
Zn insoluble per gram Fe, grams	0.0525	0.094	0.125	0.113	0.104

TABLE 6.—*Results of Roasting $ZnS + 2FeS_2$ with Increasing Amount of ZnS up to* 100 *Per Cent. Excess of That Required by Ratio Zn*: 2 *Fe, Walsh*[a]

Roasted 8 Hr. at 600° C. Rabbled Every 10 Min.
Reagents: Blende Zn = 58.23 Per Cent.
Pyrite Fe = 46.20 Per Cent.

Sample Number	53	54	55	56	57	58
Blende, grams	3.40	4.08	4.76	5.44	6.12	6.80
Pyrite, grams	7.20	7.20	7.20	7.20	7.20	7.20
Zn in mixture, grams	1.98	2.38	2.77	3.17	3.57	3.96
Fe in mixture, grams	3.32	3.32	3.32	3.32	3.32	3.32
Excess Zn, per cent	2.30	23.0	43.0	63.80	84.60	105.00
Total Zn, assay per cent	23.30	25.90	27.95	29.80	31.80	32.70
NH_4Cl sol. Zn, assay per cent	20.7	23.6	25.6	27.20	28.00	33.40
Zn solubility in NH_4Cl sol	88.80	91.18	91.70	91.40	88.1	87.8
Insol. Zn determined, grams	0.22	0.21	0.23	0.27	0.43	0.49
Insol. Zn (theoretic), grams[b]	1.935	1.935	1.935	1.935	1.935	1.935
Ferrite formed, per cent	11.36	10.85	11.88	13.95	22.20	25.30
Loss of Zn solubility per unit of Fe	0.287	0.248	0.248	0.275	0.403	0.435
Zn insol. per gram Fe, grams	0.066	0.063	0.069	0.081	0.195	0.147

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[a] In this series there is observed the contemporaneous oxidation of the sulfides of zinc and iron. The amount of ferrite formed was low, due to the influence of SO_2 generated by both minerals. The effect of SO_2 on ferrite is that of decomposition, not merely retardation.

[b] 3.32 g. Fe × 0.584 = 1.935 g. Zn theoretically insoluble.

TABLE 7.—*Results of Roasting* $ZnS + 2FeS_2$ *with Increasing Amount of* ZnS *up to* 100 *Per Cent. Excess of That Required by Ratio* $Zn : 2\ Fe$[a]

Roasted 8 Hr. at 700° C. Rabbled Every 10 Min.
Reagents: Blende Zn = 58.23 Per Cent.
Pyrite Fe = 46.2 Per Cent.

Sample Number	35	36	37	38	39	40
Blende, grams	3.40	4.08	4.76	5.44	6.12	6.80
Pyrite, grams	7.20	7.20	7.20	7.20	7.20	7.20
Zn in mixture, grams	1.98	2.38	2.77	3.17	3.57	3.96
Fe in mixture, grams	3.32	3.32	3.32	3.32	3.32	3.32
Excess Zn, per cent	2.30	23.0	43.1	63.80	84.6	105.0
Total Zn, assay per cent	25.40	28.40	31.10	33.60	35.20	37.20
NH_4Cl Sol. Zn, assay per cent	18.90	19.25	20.00	24.90	27.00	27.10
Fe, assay per cent	42.55	39.65	37.30	34.90	32.70	31.40
Zn solubility in NH_4Cl sol.	74.50	67.80	64.40	74.60	76.70	72.90
Insol. Zn, determined, grams	0.51	0.77	1.01	0.80	0.83	1.08
Insol. Zn (theoretic), grams[b]	1.935	1.935	1.935	1.935	1.935	1.935
Ferrite formed, per cent	26.35	39.80	52.20	41.30	42.90	55.80
Zn insoluble per gm. Fe grams	0.153	0.231	0.304	0.241	0.250	0.325

[a] This series is identical with Series 9 except that the roasting temperature was maintained at 700° C. instead of 600° C. The effect of 100° rise of temperature is clearly seen, yielding ferrite in excess of 100 per cent. over the figure obtained when a temperature of 600° C. was maintained. The solubility also falls materially.

[b] 3.32 g. Fe × 0.584 = 1.935 g. Zn theoretically insoluble.

the zinc is present as marmatite plus the usual admixtures, including free pyrite.

Series 3, *Table* 3.—$ZnO.2FeS_2$, 625° C. In this test the percentage of ferrite falls scarcely at all, with increase slightly and uniformly with increase of iron (see Fig. 3).

Series 4, *Table* 4.—Ferrite falls uniformly.

Series 5, *Table* 5.—$ZnS.2FeS_2$, 625° C. There is a much smaller amount of ferrite formed in roasting this mixture, because of the large amount of SO_2, which inhibits or decomposes the ferrite. The zinc solubility is quite high for the zero excess of iron, falling to a constant of 64 per cent. for 80 to 100 per cent. excess iron (see Fig. 5).

Series 6, *Table* 6.—$ZnS.FeS_2$, 600° C. The characteristics of this test are similar to those of Series 5 except that the blende is increased to 100 per cent. excess instead of the pyrite and the temperature maintained at 600° C. instead of 625°. The results show an increase of ferrite for each increase in ZnS.

Series 7, *Table* 7.—$Zn.2FeS_2$, 700° C. The characteristics of this test are the same as Series 6 except for temperature, which was maintained at 700° C. There is a decided lowering in the zinc solubility and elevation in the percentage of ferrite formed (see Fig. 7).

TABLE 8A.—*Results of Roasting Blende with Very Small Amount of Pyrite Progressively Increased*

Roasted 8 Hr. at 625° C. Rabbled at 10-min. Intervals

Reagents: Blende { Fe = 0.6 Per Cent.
Zn = 58.23 Per Cent.

Pyrite Fe = 44.40 Per Cent.

Sample Number	1	2	3	4	5	6
Blende, grams	10.0	10.0	10.0	10.0	10.0	10.0
Pyrite, grams	0	0.2	0.4	0.6	0.8	1.0
Zn in mixture, grams	5.823	5.823	5.823	5.823	5.823	5.823
Fe originally present in blende, grams	0.0600	0.0600	0.0600	0.0600	0.0600	0.0600
Total Fe in calcine grams	0.0600	0.1488	0.2376	0.3264	0.4152	0.504
Zn in calcine, assay per cent	63.70	62.50	61.20	58.80	56.80	53.80
NH_4Cl sol. Zn in calcine, assay per cent	62.80	61.40	60.00	57.40	55.30	52.20
Fe in calcine, assay per cent	0.65	1.52	2.02	3.29	4.05	4.66
Zn solubility in NH_4Cl sol	98.65	98.35	98.05	97.70	97.40	97.10
Zinc insol. in NH_4Cl determined, grams	0.0786	0.0960	0.1135	0.1338	0.1514	0.1687
Zn insol. (theoretic) (grams Fe × 0.584), grams	0.0351	0.0870	0.1386	0.1905	0.2425	0.2940
Ferrite formed (theoretic), per cent	224.00	110	81.90	70.40	62.50	57.40
Zinc insol. (NH_4Cl) per gram Fe, grams	1.310	0.645	0.478	0.410	0.365	0.335

Series 8A and B, Tables 8A and 8B.—Blende and pyrite in great variation of the latter from small to large amount were mixed and roasted 8 hr. at 625° C. The results show that for a very small amount of Fe the ferrite formation is greatest. This gradually recedes for increased amount of FeS_2. With extremely small amounts of FeS_2 there is indicated a compound $2ZnO.Fe_2O_3$ under certain ratios (see Fig. 8).

TABLE 8*B*.—*Results of Roasting Blende with Very Small Amount of Pyrite Progressively Increased*

Roasted 8 Hr. at 625° C. Rabbled Every 10 Min.

Reagents: Blende Fe = 0.6 Per Cent.
Zn = 58.23 Per Cent.
Pyrite Fe = 44.40 Per Cent.

Sample Number	8	9	10	11	12	13
Blende, grams	7.0	7.0	7.0	7.0	7.0	7.0
Pyrite, grams	0.875	1.050	1.225	1.400	1.750	2.1
Zinc in mixture, grams	4.075	4.075	4.075	4.075	4.075	4.076
Fe in blende, grams	0.0420	0.0420	0.0420	0.0420	0.0420	0.042
Fe in pyrite, grams	0.3885	0.4662	0.5440	0.6220	0.7775	0.933
Total Fe in calcine, grams	0.4305	0.5082	0.5860	0.6640	0.8195	0.975
Total Zn in calcine, assay per cent	52.90	48.90	48.60	47.10	46.40	46.00
NH_4Cl sol. Zn in calcine, assay per cent	50.59	47.21	46.75	45.15	44.35	43.7
Fe in calcine, assay per cent	5.58	6.03	6.98	7.67	9.32	11.0
Zn solubility in NH_4Cl sol.	96.80	96.60	96.30	95.70	95.58	95.1
Zn insol. in NH_4Cl (determined), grams	0.1304	0.1385	0.1527	0.1662	0.1800	0.19
Zn insol. (theoretic) (grams Fe × 0.584), grams	0.2515	0.2970	0.3421	0.3880	0.4770	0.570
Ferrite formed, per cent. of theoretic	51.80	46.65	44.46	42.85	37.78	35.00
Zn insol. (NH_4Cl) per gram Fe, gram	0.303	0.2725	0.2608	0.2505	0.2200	0.204

DISCUSSION

U. C. TAINTON, Kellogg, Idaho.—I would like to ask Mr. Hanley if he has made any tests on the solubility of ferrites as affected by temperature. I have been coming in contact with that in a rather pronounced way lately. We have been roasting an ore in which the zinc is present entirely as marmatite. All of the iron is already combined with zinc and in the roast it has been difficult to avoid a local high temperature when the ore drops from one hearth to another. I believe that is the reason for the result that we have observed, that from this particular ore the ferrite is most unusually insoluble. A treatment of 1 hr. with 10 per cent. acid makes very little difference in it. We have had to lengthen our leach to 3 hr. and use 15 per cent. acid in order to effect a substantial decomposition. I believe the reason of that is the local high temperature in the furnaces. Has Mr. Hanley any information that would bear on that subject?

H. R. HANLEY.—No, I have not, as far as any detailed statement, except that the high acidities will usually affect the solution of ferrites which are normally quite insoluble in weak acid. I have made no specific tests except here and there to show that these compounds which were normally insoluble in weak acid would be soluble in high acid—say above 15 per cent.

U. C. Tainton.—We are using the magnetic separation at Kellogg, and owing to the fact that all of our zinc carries iron, we find it possible to pick up practically all of the zinc if we make the field sufficiently strong. In fact, one of the men on the roaster floor became a little too zealous one day and increased his field to the limit and picked up practically everything off the belt, so that there was no oxide portion at all. Everything went into the ferrite portion.

We find that the solubility of the ferrite is fairly definitely a function of its magnetic susceptibility. In other words, the more magnetic a ferrite is the more difficult it is to dissolve in acid. As you get toward the less susceptible ferrite, it is fairly easily decomposed with, say, 6 or 8 per cent. acid, but the highly magnetic part of the ferrite is often hardly to be broken down with long agitation with strong acid of 15 per cent. or more.

E. L. Jorgensen, Irvington, N. J.—I am not experienced in zinc roasting, but I have had some experience in copper processes. Great similarity seems to exist.

About 15 years ago, at the plant of the Braden Copper Co. in Chile, we started our roasting with table concentrate and obtained certain specific results as to copper ferrites corresponding to the zinc ferrites. With the advent of oil flotation the grain size of the concentrate materially changed. We expected improvements in the roasting of the finer grains, but found the opposite to be the case. The difficultly soluble copper increased and more iron had to be dissolved to obtain a good extraction of copper.

This extra iron came from ferrites that had to be decomposed. Sizing tests of the roasted material showed that ferrites increased with the fineness of the material. Under the microscope the finer particles showed complete fusion.

To improve results we had to decrease the temperature of roasting, reduce the speed of the rabble arms and increase the thickness of the bed of material in the roaster.

These results suggested, I think, the consideration of separate roasting of different grain sizes where conditions would permit it.

U. C. Tainton.—We made quite a number of investigations of the solubility of the ferrites in zinc ores, leaching out the copper compounds with ammoniacal ammonium chloride until no more copper was soluble, and then we could regard the remainder of the copper as ferrite. We found, by drawing a curve of the solubility of that copper in increasing strength of acid, that the curve was much steeper than in the case of zinc ferrite. Copper ferrite would give a very low extraction with an acid strength of, say, 2 per cent. I think about 30 per cent. extraction would be the starting point with 2 per cent. acid, and by the time the acid strength was increased to 10 per cent., it would raise the extraction to 99.9, or practically total solubility; but in the case of zinc where the solubility might be perhaps 78 per cent. with a 2 per cent. acid, it would only increase to about 92 per cent. with 20 per cent. acid.

R. M. Roosevelt, New York, N. Y.—I have not heard anyone mention this paper as applying to cadmium. Cadmium recoveries at present are all the way from 80 and 85 to 90 per cent.

U. C. Tainton.—We have not made any actual investigations of the formation of cadmium ferrite, although I imagine that it probably is formed, but the solubility of the cadmium seems to follow very closely that of the zinc. Its behavior as regards increasing acid strength and as regards the effect of temperature seems to be very closely parallel to that of zinc.

H. R. Hanley.—The solubility of cadmium in the Joplin field in the ores containing around 0.3 to 0.35 per cent. cadmium is about 85 per cent., roasting around temperatures of 675°.

C. S. Witherell, New York, N. Y.—Mr. Tainton touches on the question of the increase in temperature of roasting at the points of fall of the ore, particularly its effect on the fine ores. At such points you may obtain a temperature that approaches the theoretical temperature of combustion; the temperature of fine particles may be increased to nearly 1000° C., whereas the normal temperature of the furnace is probably around 600° or 700° C., where the object is to keep down the formation of ferrite, or rather ferrate. I remember that Thomas French gave us information on that same point some years ago.[2]

U. C. Tainton.—Zinc ferrite, if fused in an electric furnace, becomes insoluble in every acid known, including hydrofluoric. In these furnaces with as many drop holes as we are making nowadays the ore gets much more suspension roasting than it used to get in the old furnaces with a few drop holes, and each particle has a much greater chance of flashing than it used to have.

We are converting one of our roasters at Kellogg now to a down-draft roaster with the idea of decomposing ferrites with the SO_2 and then adding sulfite to the lower hearths to obtain the roast reaction, decomposition of the sulfate by the sulfite and so bring back to oxides. That, of course is not our idea. That was developed in the furnace, and we are trying it out.

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[2] T. French: Roasting Zinc Ores for Electrolytic Reduction. *Eng. & Min. Jnl.* (1922) **113**, 286.

Blast Roasting at Cerro de Pasco

By Glenn A. Keep,* La Fundicion, Peru

(New York Meeting, February, 1929)

This paper is not an announcement of the successful conclusion of the Cerro de Pasco Copper Corporation's pacos-pyrite problems, but merely a description of the commercial-scale, intermittent-roasting operations conducted by the writer, J. H. Lucas and others of the staff. It was prepared by request, for presentation at the February meeting of the Institute.

The results obtained in Peru confirm the 1-ton-a-day experiments completed about three years ago at Kearny, N. J. by C. C. Gordon and Mr. Lucas, where it was first demonstrated that a mixture of about 80 per cent. pacos ore with 10 per cent. pyrite ore and 10 per cent. salt could be roasted intermittently in a small 1-ton-a-day Holt-Dern furnace without extraneous fuel, producing porous calcine that could be leached rapidly. Mr. Keep and his staff, by successfully introducing improvements in the construction of the commercial furnace and in their handling of material, have established beyond doubt a low cost for roasting. The other steps in the process of treating pacos and pyrite ores have not as yet been demonstrated as sufficiently conclusive to warrant their acceptance for a permanent plant.

Cerro de Pasco Ores

The ores of the Cerro de Pasco Copper Corpn. at Cerro de Pasco, Peru, are situated at an altitude of 14,000 ft. The town and mines are served by the Cerro de Pasco Railroad, which operates between Cerro de Pasco and Oroya, the site of the company's copper and lead smelters.

Table 1.—*Average Analysis of Cerro de Pasco Ores*

	Pacos Ore	Pyrite Ore
Silver, oz.	10.3	12.0
Gold, oz.	0.007	0.01
Lead, per cent.	1.7	1.8
Copper, per cent.	0.19	0.27
Zinc, per cent.	1.4	1.02
Iron, per cent.	13.4	34.45
Silica, per cent.	61.8	24.5
Alumina, per cent.	2.5	0.62
Lime, per cent.	1.5	0.20
Manganese, per cent.	0.42	0.09
Sulfur, per cent.	1.1	34.2

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[*] Superintendent, Pacos Pyrite Plant, Cerro de Pasco Copper Corpn.

The low-grade ores under consideration can be divided into two classes, namely, pacos ore and pyrite ore. The pacos ore is an oxidized surface capping, which overlies the unaltered sulfide or pyrite ore. The orebodies have been extensively developed by years of mining work supplemented by drilling. Many millions of tons await the advent of a successful treatment process. Average analysis of the Cerro low-grade ores is given in Table 1.

Metallurgical Investigations

During the past 5 years, efforts to solve the metallurgical problem presented by the Cerro low-grade ores have been concentrated on a chloridizing roast of the crushed ores followed by leaching with a brine solution. This work has involved the economical extraction of silver, copper, lead and bismuth contained in the pacos and pyrite ores. Covering a period of 2 years, a 1-ton test plant was operated in New Jersey with inviting results and subsequently a pilot mill utilizing commercial-sized units was constructed in Peru, with the object of further perfecting the process and at the same time of obtaining thorough and reliable data pertaining to metal yields and costs.

Early experiments indicated good recoveries when the ores were chloridized in blast-roasting furnaces, using the sulfur in the pyrite ore as fuel rather than costly coal, oil, or gas. The thermal efficiency of a blast roaster is high because the air needed for combustion of the fuel is preheated in passing through the hot calcine before reaching the roasting zone and the charge above the roasting zone is heated by the hot gases leaving the roasting zone. In blast roasting the heat is generated in the charge itself and is uniformly distributed over the roasting zone. This permits very close temperature control and uniform operating results. The low temperature at which the gases leave the charge in blast roasting reduces volatilization losses to a minimum. The simple and relatively cheap construction of blast roasters and the low indicated operating and repair costs made the blast roasting in Holt-Dern type roasters very promising.

Construction of Roasters

In previous installations, Holt-Dern roasters have been constructed wholly of concrete, but after due consideration it was decided to employ steel in erection of the pilot plant roasters. A general conception of the appearance and construction of these furnaces can be gained from Fig. 1. The roaster consists essentially of four components: an overhead charge bin, a roasting zone, a calcine-receiving hopper and the grates and drive mechanism.

The design and construction of the roaster grates together with their driving machinery are the most important features of the Holt-Dern type roaster. Mechanical action of the driving mechanism is diagrammatically presented in Fig. 2.

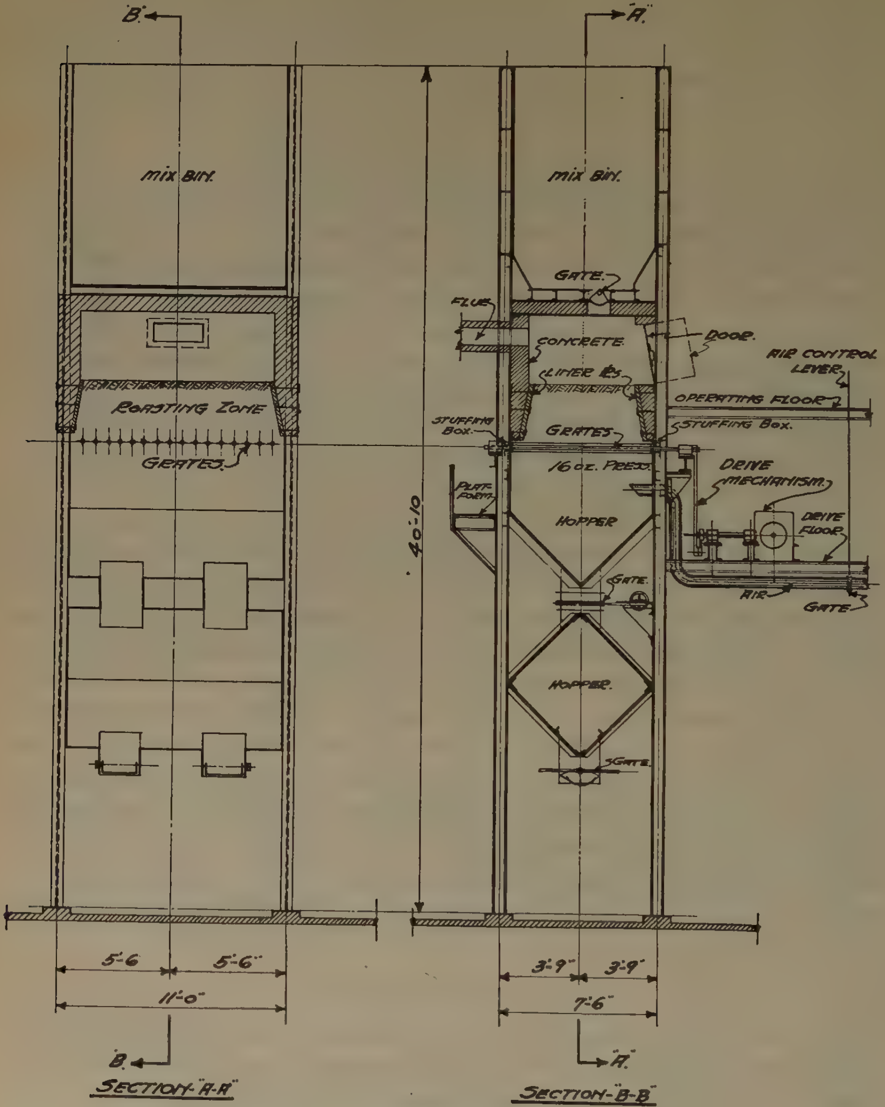


FIG. 1.—SECTIONS THROUGH ROASTER.

When operations were started in the pilot mill, the rocking motion of the grates was insufficient to afford a quick discharge on the rather hard calcine. By increasing the length of the drive cranks, the grate throw was increased to 80°, which insures a total time of discharging and reloading of 15 to 20 min. Augmenting the grate throw resulted in discharge of larger calcine lumps varying from 3 to 4 in. dia. However, these lumps were leached without difficulty, with satisfactory metal recoveries.

Construction of the cast-iron grate bars is shown in Fig. 3. The old style bars have teeth on both sides with solid ribs at top and bottom. When the top rib is worn, the bars are reversed so as to bring the bottom rib into action. When both ribs are about one-half worn away, the grate bars are recast into new ones.

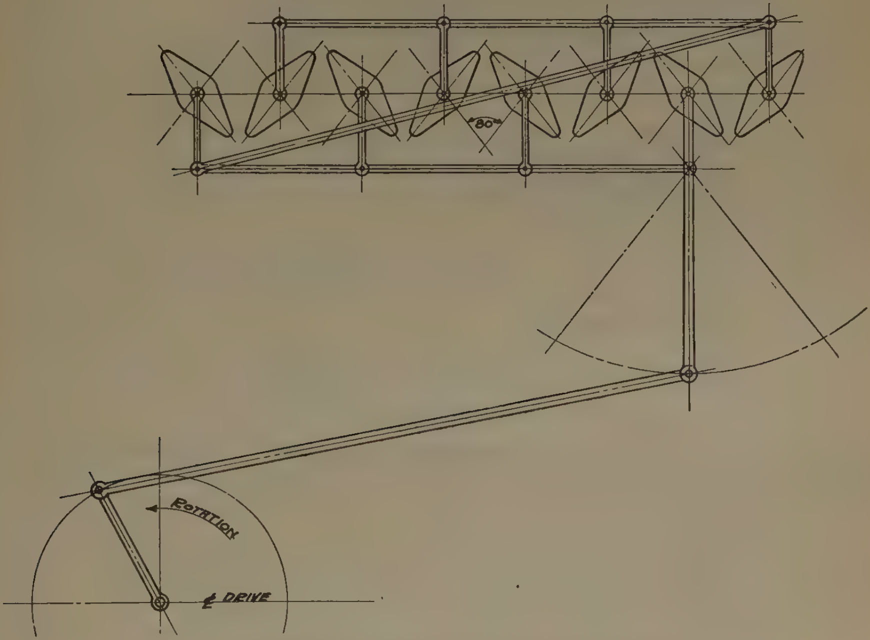
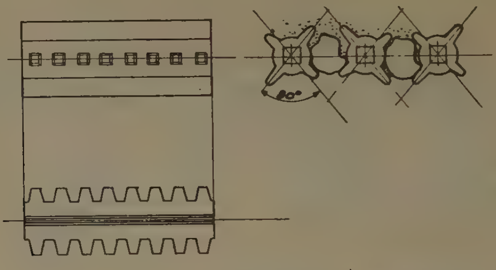


Fig. 2.—Side elevation of drive mechanism.

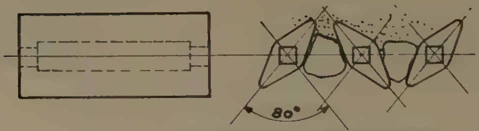
The action of the rocking grates causes the roasted charge to heave and break into large pieces. Caught between the grate ribs, the calcine is then crushed and passes into the receiving hopper from which it can be removed to suit the convenience of the operator.

As shown in Fig. 3, the construction and action of the old style grate make it possible for calcine lumps of certain sizes to ride between the grate bars. They are not crushed by the rocking grates and are still too large for passage through the discharge openings. A new grate bar, indicated in Fig. 3, has been designed to obviate riding lumps and to secure more rapid discharge. The new bars are spaced in the same way as the old bars, but having no side teeth they will discharge lumps of larger size, which we judge can be leached without difficulty. It is expected that the new bars will save labor and also reduce the time of discharging, with a resulting small gain in roaster capacity.

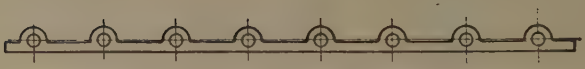
To avoid a smoke nuisance on the grate-level floor, it is necessary to avoid gas leakage around the grate shafts. Generally, this type of roaster has been provided with a partial stuffing box at the points where the shafts pass through the roaster sides. The reciprocating crank arms have been mounted directly against these stuffing boxes so that tightening of the stuffing gland was not possible. Also, in this design the stuffing boxes on the drive side of the furnace acted as bearings taking the thrust of the drive mechanism. This driving thrust entailed considerable wear on the combined stuffing-box bearing and as the gland could not be tightened, leakage of gas ensued.



OLD STYLE GRATE BAR



PROPOSED NEW GRATE BAR



OUTBOARD THRUST BEARING BLOCK

FIG. 3.

It appeared that this arrangement was subject to improvement. Hence the pilot plant roasters were provided with standard stuffing boxes and glands, which can be tightened as occasion demands. The thrust from the driving mechanism was absorbed by outboard individual bearings. Dealing with rather hard calcine, experience soon demonstrated that it was difficult to hold these bearings in place. Accordingly the individual bearings were replaced by a cast-steel outboard thrust-bearing block, as shown in Fig. 3. The present roaster design is successful in preventing gas leakage and also handles the heavy driving thrust in a satisfactory manner.

Next to the grates in importance is the size, depth and shape of the roasting zone. In the original installation, roaster sides were given a slope of 1 in., making the cross-section at the grates 2 in. larger than at the top of the roasting zone.

Using only pyrite as fuel in the mix, the resultant calcine is harder than usually obtained in Holt-Dern roasters. Furthermore, during roasting the charge swells and presses out against the roaster sides. Consequently, on starting the grates, the bottom of the roasted calcine body was undercut, while the main body of the calcines hung up and refused to drop down on the rocking grates. Considerable labor was then required to cut a channel through the roasted calcine, and in this way force it to drop on to the grates. This difficulty was corrected by materially increasing the angle of slope on the sides of the roasting zone. This procedure reduced the volume of this zone and, to compensate, the depth was slightly increased. Present inside dimensions at the top are 5 ft. 4 in. by 9 ft. 5½ in. and 6 ft. 2½ in. by 10 ft. 2½ in. at the grate level with a vertical depth of 33 inches.

Generally, the sides of the roasting zone in Holt-Dern roasters have been constructed of concrete, and it has been necessary to reline such roasters at intervals of 6 months to 1 year, with resultant expense and loss of roasting capacity. When side slope was increased in our roasters, cast-iron liner plates ¾ in. thick were installed on the sides by bolting them over the concrete lining. These plates show practically no wear or corrosion and indications point to a life of many years. Our roasters have never been relined. Since the side slope has been increased, roasted charges do not hang up in the roasting zone after the grates are put in action.

The roasting zone in our furnace is 33 in. deep. In order to determine the effect of roasting a deeper bed of charge, one roaster was remodeled to take a 45-in. bed. The operation of this roaster demonstrated that it requires proportionally more time to roast the 45-in. bed than it did the 33-in. bed and the tonnage handled per roaster day was considerably less with a 45-in. bed than with a 33-in. bed. More time and labor are required to discharge the roaster when operating with 45-in. bed than with 33-in. bed.

Two calcine-receiving hoppers were installed for our special purposes. Ordinarily only one calcine hopper of adequate size is necessary. However, this hopper should be provided with airtight discharge gates in order to prevent gas leakage.

OPERATION OF FURNACE

The modus operandi covering operations of a blast-roasting furnace for chloridization may be briefly described as follows:

1. Mix of properly crushed ore is made with salt, water and fuel, either coal or sulfide ore. Fuel content should produce a resultant roasting temperature of 650° to 700° C. Addition of correct amount of moisture is very important in yielding a porous mix which will allow passage of air through the charge.

2. In starting the roaster, a bed of kindling wood about 10 in. deep is spread over the roaster grates and ignited. A small amount of air is turned under the grates and the wood is allowed to burn to a mass of glowing coals. Then some mix is shoveled on to the hot coals, more air furnished, and in this way a bed of roasting ore about 6 in.deep is obtained. From the overhead mix bin, a foot of green mix is drawn into the furnace; and so on, until the roaster is filled.

3. When the charge is roasted through on top, air is shut off, roaster grates are started in action and the charge dropped about 22 to 24 inches.

4. Hot calcines 12 to 15 in. deep remain on the grates. After spreading the red hot calcine evenly over the grate area, a full charge of mix is drawn into the roaster from the overhead bin. The new charge is leveled on top and air pressure turned back on the roaster.

5. The new charge is ignited from the hot bottom layer and proceeds to roast through to the top, thus completing one cycle.

Nine to ten cycles per day are made on our roasters. Charts from a recording air-pressure gage clearly indicate the number of cycles, air pressure and time consumed in discharging and reloading the roaster between cycles.

Importance of Good Mixing

Holt-Dern roasters will function very well when provided with properly mixed charges, hence the secret of successful roaster operation may be said to lie in the mixing department. Poorly mixed and proportioned charges will cause all sorts of trouble in the roaster; insufficient fuel results in low temperatures with the chance that the roast may die out or speed of roasting be reduced; a high fuel ratio may cause the charge to fuse and sinter, in which case it will require much barring in order to make the sintered calcine discharge through the grates and, moreover, values in sintered calcine will not be fully recovered in subsequent leaching operations; porosity of the charge as determined by the moisture content must be correct or the roasting speed will be reduced and a severe irregularity in this respect may result in loss of the fire.

In view of this situation, much consideration was given to the design of mixing equipment which would insure uniform roaster charges at all times. Ores are crushed dry to pass an 8-mesh screen with 0.084-in. square openings, and after passing automatic samplers, the fine-crushed materials are stored in overhead steel cylindrical bins. There are three bins for pacos ore, two bins for pyrite ore, and one bin for salt. Assays are obtained upon all materials in these bins before the batch weights are computed.

The batch for the mixer is drawn from the proper bins into a traveling weight car, which discharges into a concrete mixer of 1 cu. yd. capacity, fitted with dust doors. The batch is mixed dry for 1 min., then a weighted amount of water or weak mill solution is run into the mixer. After further mixing to incorporate the moisture uniformly the batch is discharged into a skip and hoisted to the roaster mix bins.

The damp mix will not slide on a 45° angle and tends to stick and build up in the buckets of an elevator. The skip handles the finished mix without any operating troubles.

It was found that the pacos ore varied from 0.5 to 3.5 per cent. sulfide-sulfur when handled in lots of 90 tons. This variation from bin to bin required the figuring of charges for each lot of pacos ore. This was not serious, but variations of sulfide-sulfur from bottom to top of the same bin could not accurately be considered in computation of roaster charges. This condition was not fully satisfactory, and so, for over a year, receipts of pacos ore have been made into 3000-ton ore beds. In this way pacos ore for the mill is obtained which does not vary greatly in its fuel value.

The blast roasters are operated with pyrite only as fuel. The optimum mix for our conditions is as follows: sulfide-sulfur, 4.9 per cent.; moisture, 7.0 per cent. With this fuel and moisture content, the charge will roast at a temperature of 650° to 700° C., when furnished with an air pressure of 16 oz. Variations of 4.7 to 5.2 per cent. sulfide-sulfur, and 6.8 to 7.3 per cent. moisture are permissible, and it is easily practical to hold the mix within these limits.

Roaster Capacity

In previous Holt-Dern roaster installations, the average daily capacity of one furnace has been 22 to 25 tons of ore. Our furnaces will average 38 tons of dry ore per day and on several occasions the monthly average has been 40 tons or over per day.

One reason for this increased roaster capacity lies in the fact that Holt-Dern equipment has always functioned on air pressures of 8 to 9 oz. while our furnaces operate with 16-oz. pressure. It was considered that pressures higher than 8 or 9 oz. were not practical because of the tendency for blowholes to form in the roasting zone. However, with a carefully designed mixing department to insure uniform fuel value and porosity in the mix charge, our experience indicates that blast roasting can be accomplished at 16-oz. pressure without formation of blowholes or other operating troubles. Air is furnished by a General Electric constant-pressure centrifugal blower operating at 3600 r.p.m.

While it has been common practice to blast roast a charge containing fine coal as one constituent of the mix, our fuel is sulfide-sulfur only, and observations lead us to believe that the speed of roasting is faster when sulfide-sulfur alone is used as fuel. Attention has already been called to the fact that Cerro blast-roasted calcine is unusually hard. Experiments have been conducted with the view of producing a softer product. Fine coal was substituted as part of the necessary fuel and the amount of sulfide-sulfur was correspondingly decreased. The resultant calcine was much softer and the time of discharging was reduced with less labor, but the speed of roasting was not as fast, and there was a decrease in roaster tonnage. Furthermore, it is desirable in our case to utilize pyrite ore as fuel, not only to save the cost of coal but also to recover values from the Cerro low-grade pyrite ore.

An attempt was made to obtain softer calcine by reducing the moisture content of the mix. As the moisture content was gradually reduced there was retardation in the rapidity of roasting, and finally the green mix over the roasting ore became very wet from condensed vapor. These rather dry mixes acted in the same manner when roasted in the laboratory roaster. Explanation of this rather peculiar occurrence does not seem difficult. Porosity of the mix is reduced, hence less air passes through the roaster charge and consequently there is loss in roasting speed. The quantity of air can thus be curtailed until it no longer has the capacity to evaporate and carry away the moisture expelled by roasting. Obviously, dry mixes were not a solution of the hard calcine problem.

Control of Operating Costs

It is obvious that any means of decreasing the time and labor required for discharging a roaster will be reflected in lower operating costs. Production of a softer calcine would effect this result, but since the use of pyrite ore as fuel yields a hard calcine it is believed that the desired condition can be obtained by discharging larger lumps through the grates by the use of an improved grate box, as illustrated in Fig. 3. When chloridizing in a blast roaster, only a small percentage of the chlorine contained in the mix is eliminated during the roasting operation. Covering operations of 1928, the chlorine of the mix averaged 4.89 per cent. while the roaster calcines averaged 4.27 per cent. chlorine. The difference, or chlorine lost in the roasters, was 0.62 per cent.

Chlorine contained in the roaster calcines is dissolved in the leaching tanks, consequently the volume of salt solution increases in the mill circuit. As it is not economical to discharge waste salt solution, other means to control the volume of mill solution have been devised. Percentage of salt on the basis of dry ore in the mix has been reduced from 9 to 7 per cent. salt (88 per cent. NaCl) without any detrimental effect on subsequent metal recoveries. Weak wash solutions obtained as the last effluent when leach tanks are washed at the end of the leaching cycle have been segregated. This weak solution constitutes all wash effluents from 15° to 3° Bé. and is utilized for moistening the mix, instead of water. Salt equivalent to that contained in the weak solution is deducted from the amount of fresh salt added in the mixer charge. By this procedure it is practical to control solution volume without waste.

On account of the sulfide-sulfur content of pacos and pyrite, the proportion of each in the mix to give proper roasting conditions and resultant calcine is in the ratio of 88 per cent. pacos to 12 per cent. pyrite. As the ore deposits occur in a different ratio, large-scale experiments have been conducted in the mill with the object of treating a greater proportion of pyrite. As previously explained, the roasted calcines contain 4.27 per cent. chlorine, which can be utilized for chloridization of more ore. Without addition of more salt, the roasted calcine is remixed with water or weak solution and with sufficient fresh pyrite to furnish the requisite fuel value in the mix, which is then "reroasted." This reroasting can be carried on three or four times until the pacos and pyrite ores are handled on a 50-50 basis. The saving in salt cost can be applied against the expense of reroasting so that the actual additional cost of this procedure is rather low.

Six leach tanks—about 350 tons of calcine reroasted as described—were treated by regular methods in the leaching department with an increase of 3 per cent. in silver extractions. Therefore, it appears that reroasting tends to give more thorough chloridization. In operation of Holt-Dern roasters, more or less green unroasted ore will be discharged with the calcine and reroasting reduces the proportion of such unroasted material in the feed to the leaching tanks.

Based on experience in the pilot mill, it is our opinion that blast roasting can be accomplished at Cerro de Pasco at a cost of 10 c. per ton of ore, exclusive of salt expense.

Acknowledgment

The author wishes to express his thanks to those of the pilot mill staff who have ably assisted in this work: J. H. Lucas, D. S. Sanders, A. A. Olsen and D. G. Headley.

DISCUSSION

U. C. Tainton, Kellogg, Idaho.—We have always supposed that a zinc concentrate roasted by a sintering method or blast roast of any sort, whether Dwight-Lloyd or Greenawalt sintering machines, or Holt-Dern roasters, would naturally give a very insoluble calcine. We expected to form a very insoluble ferrite and to form a large amount of silicate, but that opinion was upset recently.

The St. Joseph Lead Co. sent us a quantity of zinc concentrate that had been first roasted, I think, in a Wedge furnace down to about 8 per cent. sulfur, and then the roast completed on a D. & L. This material, after crushing, yielded an extraction of about 95 per cent., whereas the same concentrate in the usual treatment gave 96 per cent.; so it is evidently quite a practical thing to roast ores in that way.

I hope that eventually some means will be found to apply the Holt-Dern roaster, in the zinc business. At present zinc-roasting costs are much too high.

Government Potash Exploration in Texas and New Mexico*

By G. R. Mansfield† and W. B. Lang,† Washington, D. C.

(New York Meeting, February, 1929)

The third year of Government exploration for potash by the U. S. Geological Survey and the U. S. Bureau of Mines under the authorization of the act approved June 25, 1926 (Public 424-69th Cong.) is drawing to a close. The results thus far obtained are thought to be sufficiently interesting and satisfactory to justify the presentation of a brief preliminary account of the work.

Review of Earlier Exploration

The general geologic similarity of the Permian basin of Texas and New Mexico to the potash field of Germany had long ago suggested the possibility of the presence of potash salts somewhere in that basin, but petroleum exploration was then making little headway there and the necessary data could not be obtained. The discovery by Udden in 1912 of potash in brine from the Spur well in Dickens County, Texas, was the first direct evidence of its presence. In 1915–17 the U. S. Geological Survey drilled a test well with standard tools at Cliffside near Amarillo, but failed to find potash. Nevertheless, an observer has been kept in the field, at first in cooperation with the Texas Bureau of Economic Geology but since 1922 by the Survey alone, to secure data and samples from oil and gas tests. Since the discovery of oil in Reagan County in 1923 exploration by drilling has been rapidly extended to many parts of western Texas and southeastern New Mexico.

The identification of the potash-bearing mineral polyhalite was first announced by the Geological Survey in June, 1921, the material having come from cuttings from the Bryant well in Midland County and the Burns well in Dawson County, Texas. Successive discoveries and announcements of similar purport aroused widespread interest in potash exploration, which culminated in the action by Congress already noted.

Permian Salt Basin

The Permian salt basin extends generally northeastward from western Texas and southeastern New Mexico through the Panhandle sections of

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* Published by permission of the Director, U. S. Geological Survey.

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Texas and Oklahoma, and crossing Kansas northward includes a portion of eastern Colorado. So far as known, however, the potash-bearing salts most likely to have commercial importance are confined to the southern part of the basin. The area in which they have been found occupies about 22 counties in Texas and New Mexico and covers about 40,000 square miles.

General Stratigraphy

The formations with which the potash-bearing salts of Permian age are more directly associated are concealed throughout the basin. The mountains bordering the basin on the west and south are composed of rocks that are the correlatives of or that underlie the salt-bearing series and the same is true of the so-called Bend Arch that constitutes the eastern border. The basin in Texas and New Mexico is occupied in part by the Llano Estacado or High Plains, which is largely underlain by Cenozoic deposits. In its southward extension, the Edwards plateau, the surface rocks are chiefly Lower Cretaceous. Wells drilled in the interior of the basin may therefore pass through Cenozoic, Cretaceous (locally), Triassic and some Permian beds before reaching the saline members, whereas toward the margins all but the overlying red beds of the Permian may be absent. These stratigraphic differences, together with structural relations to be noted presently, cause the depth of the "top of the salts" to vary in different parts of the basin from 300 ft. to more than 2000 ft. The thickness of the salt series (salt and associated beds) in the Means well in Loving County totaled more than 3500 ft., though in general over an area comprising hundreds of square miles it approximates 1000 ft. The thickness of the Permian strata in the regions bordering the basin, where they pass beneath the younger rocks that form the surface of the High Plains, ranges from about 4000 to 10,900 ft. The rocks above the salt beds within the basin consist chiefly of limestones, sandstones and red sands and shales with some gravelly or conglomeratic material, but those beneath the salt series, which also includes anhydrite in the lower part, are chiefly limestones.

The formations on the east and west sides of the basin have been studied and classified by numerous geologists, but the two areas of exposure are 200 miles or more apart, the sequence of formations is not the same, and agreement regarding their correlation has not been reached. Within the salt series itself are many beds of anhydrite and common salt with which are interbedded thinner layers and lenses of potash-bearing salts. At some localities it seems feasible to match beds of possible economic importance in wells a few miles apart. Elsewhere it is known that these beds thicken and thin or disappear within relatively short distances, so that sequences in neighboring wells differ and it may be difficult if not impossible to determine the correspondence of beds.

Further details of the stratigraphy of this region are given in papers by Hoots,[1] Edwards,[2] Sellards,[3] and others.

General Structure

The Permian salt basin is a geosyncline, which still retains its general synclinal form. There were, however, crustal movements affecting land areas to the east during the epoch in which the salt beds were being deposited, for extensive wedges of red clastic material extend westward from the eastern margin far out into the basin. These are separated from each other by beds of dolomitic limestone and blue shale that die out eastward. Minor movements within the basin during deposition are also suggested by these stratigraphic differences. After the Permian beds were laid down the region was subjected to further disturbances, which permitted partial erosion of these beds and the deposition of sediments of later age upon the gently beveled edges of the earlier rocks. Such disturbances were repeated at least twice before the present structural status of the region was assumed. The structure of the overlying beds therefore differs from that of the Permian itself. Moreover, there are structural irregularities in the floor of the basin, some doubtless induced by the movements mentioned and others which perhaps are original and may have constituted barriers sufficiently persistent to have played a part in the localization of evaporation pans in which salts of various kinds were deposited. Such a barrier may have been responsible for the apparent greater abundance of chloride salts of potassium in southeastern New Mexico than in Texas. From present evidence the two regions seem to have been subjected to variations in deposition that differ in degree.

Structural Details

In Germany the potash salts that are mined are largely associated with salt domes, and salt domes have recently been recognized in Alsace, but in southwest Texas and southeastern New Mexico none have thus far been discovered. Salt domes are important structural features of the Gulf coastal region of Louisiana and Texas, and one of these in Matagorda County, Texas, has yielded potash salts at a depth of 4800 feet.

The principal structural features that have thus far had a bearing on the location of drilling sites during the present investigation lie in two zones of uplift, called by Denison[4] the Hendricks-Yates fold and the McCamey-Connell fold. The first of these extends from Lea County, New Mexico, southward and southeastward into Crockett County, Texas, and the second from southwestern Ector County, Texas, southeastward into northwestern Crockett County (see Fig. 1). Although the salt beds usually rise toward the surface eastward and westward from the depths of the great synclinal basin, the zones just outlined, which lie in the western half, seem to represent arches or anticlinal folds, for the depth of the salts increases on both sides of them. The McCamey-Connell fold has attracted more attention than the other in this investigation because of its earlier discovery and because of the relatively abundant potash data that it has furnished. Perhaps corresponding information may later be obtainable from the Hendricks-Yates fold.

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[1] H. W. Hoots: Geology of a Part of Western Texas and Southeastern New Mexico, with Special Reference to Salt and Potash. U. S. Geological Survey *Bull.* 780 (1926) 33.

[2] E. C. Edwards: Stratigraphic Position of the Big Lime of West Texas. Amer. Assoc. Petr. Geol. (1927) **11,** 721.

[3] E. H. Sellards and E. P. Shock: Core-drill Tests for Potash in Midland County, Texas. Univ. of Texas *Bull.* 2801 (1928) 159.

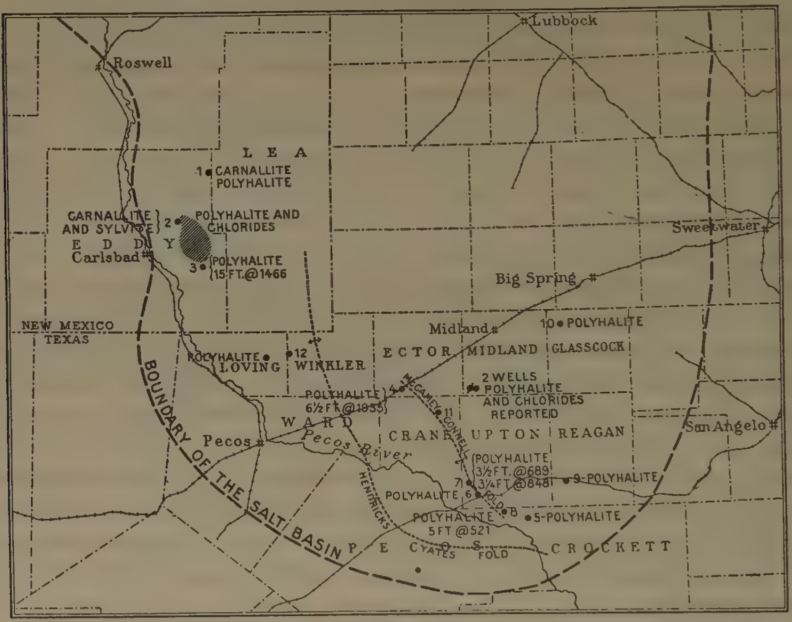

FIG. 1.—TEXAS-NEW MEXICO POTASH AREA, SHOWING LOCATION AND RESULTS OF CORE TESTS.

Conditions in New Mexico suggest that a more or less separate basin of deposition and desiccation persisted in that region. Possibly other similar basins may be found elsewhere as subsurface data become more abundant.

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[4] A. R. Denison: Personal communication. The authors are indebted to Mr. Denison for helpful criticism of this paper and for the delineation of the Texas portion of the Hendricks-Yates fold.

SALT ACCUMULATION IN THE PERMIAN BASIN

General Conditions of Accumulation

The salt series itself has thus far yielded no fossils but the rocks underlying it are definitely marine in origin and the apparent succession of the saline beds, as will be noted later, suggests deposition from marine waters rather than from inland seas not related to the ocean. It is supposed, therefore, that at the time when the salt series was being laid down the Permian salt basin was a great shallow arm of the sea that maintained at least intermittent connections with its parent body and received such inflow from the land as the arid climatic conditions of the time permitted. Evaporation from the surface of the basin in general exceeded inflow from any source for protracted periods, as indicated by the enormous extent and thickness of the salt beds. The succession of salts within the salt series indicates, however, that recurrent supplies of fresher water were available many times during the period, either through variation in climatic conditions or because of gentle crustal disturbances which enlarged or reduced the channels connecting the basin with the sea. Both sets of conditions were doubtless operative many times. Such changes altered from time to time the size and position of successive desiccation pans within the basin. Moreover, the structural relations and the saline beds in southeastern New Mexico suggest that the main basin itself may have had subordinate arms with which its connections were intermittent or occasional, and which reproduced in miniature or in a more intensive way the conditions within the main basin.

Succession of Salts

In the Permian salt basin the succession of salts is a complex problem, but certain general deductions may be made from facts already observed. First, the great extent and thickness of the beds suggests the ocean as the most probable source of the salts. Next, the succession anhydrite, polyhalite, halite and finally, in some places, more soluble minerals suggests deposition by evaporation from marine waters rather than from the waters of inclosed or entirely land-locked basins. The salts from such basins ordinarily include considerable amounts of calcium or magnesium carbonates and sulfates of sodium, calcium, and magnesium, together with borates and other salts not so commonly found in saline residues from marine waters. Again, the repetition of beds of clay, anhydrite, and halite in many of the cores indicates that the cycle of saline deposition was interrupted and renewed many times during the general period of desiccation.

Effects of Movements and Climatic Change

If the Permian salt basin may be supposed to have remained broadly stationary and the inflow of oceanic waters continuous until a final closure permitted complete desiccation of the waters, accumulation of potash salts would naturally be expected in the deepest parts of the basin, presumably centrally located, where the salts were thickest. If, however, as suggested, there were tilting or other crustal disturbances beneath the basin, the positions of the so-called salt pans holding the mother liquors, from which potash salts might crystallize out, would shift in adjustment to these disturbances. Under such conditions potash salts might or might not be found in the central portions of the basin. Again if there were marginal depressions or embayments subordinate desiccation basins might be formed in which the cycle of saline deposition might advance farther in a given time than in the main basin itself. Changes of climate would operate to check or to accelerate the cycle of deposition in the main basin or its subordinates, depending on the nature of the change, its duration, and the size of the evaporation area affected. Thus instead of the simple conditions first postulated one would expect to find numerous cycles of saline deposition in different stages of completion. Potash salts, such as polyhalite, which represent earlier stages of concentration, would be widely distributed, whereas sylvite, a product of longer continued concentration by evaporation, would be much more localized and its position would have no necessary relation to the central part of the basin. These expectations appear to be fully met. Practically every well drilled in the southern part of the salt basin encounters some polyhalite and the alternation of salts in all the cores shows that there have been many partial cycles of deposition. Sylvite, on the other hand, has been found at comparatively few localities and, so far as available records go, it is less abundant in the central part of the basin than in some marginal areas.

Secondary Changes in Salts

Mineralogical studies of the cores are bringing out interesting relationships among practically all the salts present. Anhydrite and halite appear to be primary minerals. Likewise some of the occurrences of polyhalite and sylvite indicate that they, too, occur as primary minerals in certain places. On the other hand, intergrowths of polyhalite and anhydrite, polyhalite and halite, halite and sylvite, halite and carnallite and other combinations show a wide range of replacement phenomena, which makes it fairly certain that much if not most of the potash salts present at any given locality are of secondary origin. Such a conclusion is to be expected in view of the large size of the basin and of the number and character of the movements to which it has been subjected.

CONDITIONS AFFECTING SELECTION OF DRILLING SITES

Natural Conditions

The solubility of sylvite and carnallite is so great that cuttings from the ordinary oil wells seldom show traces of either. Careful watch of drilling water, when the drill is penetrating horizons suspected of containing these chlorides may reveal sufficient changes in the composition of the water to suggest their presence but core drilling with a special chloride solution is essential in prospecting to show the number, character, and thickness of beds containing these minerals. Even under these conditions cores containing sylvite and especially carnallite are strongly etched. Polyhalite, on the other hand, is relatively insoluble and pieces of it, through slight decomposition in the presence of water, surround themselves with protective coatings of gypsum. Polyhalite is thus commonly preserved in well cuttings and affords practically the only indication that oil wells give of the presence of potash at any locality. When a number of horizons yielding polyhalite appear in the cuttings from a well, and particularly when cuttings from one or more of these horizons consist of relatively pure polyhalite, the chances of securing sylvite or carnallite at the same site seem better than when the polyhalite horizons are few and the material of poor quality.

Depth is another factor in the selection of drilling sites. In the absence of known salt domes and under the conditions already outlined there is no more reason for exploring the deeper parts of the basin in search for the more soluble potash minerals than for testing the shallower parts. With limited funds available it has seemed logical to test first those areas where polyhalite showings have been most favorable and the depth least. The McCamey-Connell fold has furnished sites for about one-half the tests. Interesting results have been obtained. On the other hand, the deeper areas are not to be neglected: three of the wells either drilled or contracted for have been located in such areas.

Although the surface generally tells little or nothing about subsurface conditions in this field, it has much to do with the practical matter of setting up and maintaining the drilling rig and its crew and with procuring and handling the supplies needed for the operation. Of two or more possible sites otherwise equally favorable that which has better transportation facilities and is nearer fuel and water will be selected. In much of the potash field fuel and water are scarce and must be hauled long distances over poor or temporary roads.

Legal Restrictions

The act of June 25, 1926, provided, among other things, that all owners or lessees of land or of mineral rights within a radius of one mile from any selected drilling site must sign contracts agreeing to reimburse the Government for the cost of exploration, whether potash were found or not, before any work at the given site could be started. The act further gave the Secretary of the Interior the right to pass upon prices that might be asked for such land if any sale or transfer were contemplated. No drilling contracts could be let until the preliminary contracts with owners or lessees were signed.

In Texas, where there is no Government-owned land, the U. S. Bureau of Mines, which assumed this part of the work, found it impossible to secure the agreements needed. Under these circumstances the Geological Survey and Bureau of Mines were automatically shut out from Texas and centered their attention on New Mexico, which still has many square miles of public land. Even here, however, the "mile radius" feature of the act proved a serious handicap and prevented the use of the best geological evidence in the location of drilling sites. Distributed among the Government lands are many tracts of state, school or railroad lands, homesteads, stock-raising homesteads, and lands affected by oil, gas, or potash permits, leaving only limited areas from which to choose. Fig. 2 shows the operation of the "mile radius" in a typical township in New Mexico.

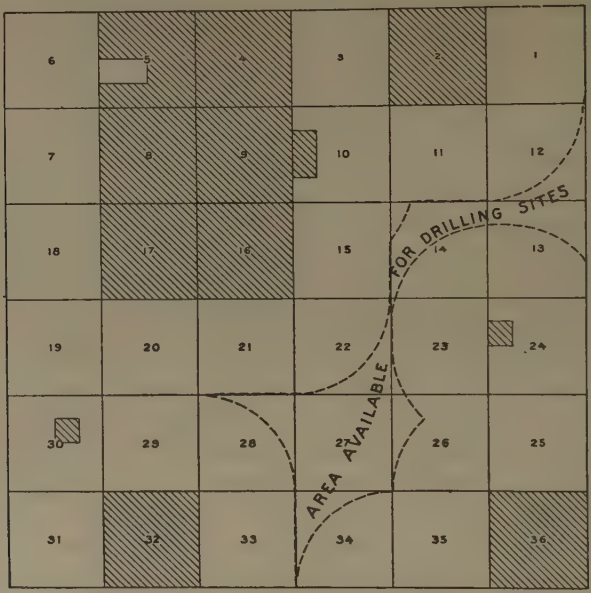

FIG. 2.—TYPICAL TOWNSHIP IN SOUTHEAST NEW MEXICO. AREA AVAILABLE FOR DRILLING SITES DOES NOT LIE "WITHIN THE RADIUS OF ONE MILE" FROM STATE AND FEE LANDS (SHADED).

As the difficulties of operation under the original act became apparent Congress passed an amendment (Public No. 759—69th Cong.) approved

March 3, 1927, which did away with the "mile radius" and other objectionable features but inserted a clause relating to oil which has hindered though it has not actually prevented the signing of contracts. This amendment made it practicable to include Texas in the exploration.

New Mexico Core Tests

Selection of Sites.—By the time the amendment was passed three sites in New Mexico had been selected by the Geological Survey and drilling at the first site was already in progress under the auspices of the Bureau of Mines. Two of the three sites were selected, subject to the limitations discussed, with the idea of finding the limits of an area in which a single core test on the McNutt permit in sec. 4, T. 21 S., R. 30 E., Eddy County, had yielded sylvite. The first site was about 22 miles to the north of the McNutt site in sec. 13, T. 17 S., R. 31 E., the second about 12 miles north-northwest of the same locality in sec. 14, T. 20 S., R. 29 E., and the third about 11 miles due south in sec. 34, T. 22 S., R. 30 E. (see Fig. 1). Geology and well data had shown that the salt beds at these localities would be relatively shallow, and so it proved. At the first test the salts were reached at 760 ft.; at the second at 335 ft.; and at the third at 365 feet.

Depth and Recovery.—Few oil wells had been drilled in that part of New Mexico at that time and stratigraphic data were meager. All three wells were therefore cored from top to bottom. The first well was 1847 ft. deep; the second 1105 ft.; and the third 1505 ft. Of the first core, 94 per cent. was recovered. The recoveries for the second and third tests were 92 and 82 per cent. respectively.

Texas Core Tests

Selection of Sites.—Oil-well drilling in western Texas has been very active in the past few years and potash in varying amounts has appeared in most of the wells from which cuttings were examined by the Geological Survey. On the whole, the McCamey-Connell fold extending from Ector to Crockett counties, as shown in Fig. 1, seemed to contain most of the wells with better showings and with shallower potash horizons. It was determined therefore to test this zone at intervals. Nine core tests have been drilled or contracted for in Texas. Of these the fourth test in southwestern Ector County, the sixth and seventh in southwestern Upton County, the fifth and eighth in northwestern Crockett County, and the eleventh in Crane County represent this structural zone. The ninth test in southwestern Reagan County and the tenth in northwestern Glasscock County are on the eastern flank of the basin. The twelfth test, in western Winkler County, is located in the trend of thickest saline deposition. Excellent showings of potash had been obtained from well cuttings at these localities.

Depth and Recovery.—At the time of writing (January, 1929) the fourth to tenth wells had been completed by the Bureau of Mines and the eleventh was drilling. A contract has been let for the Winkler County test. Of those completed the fourth is deepest, 2098 ft., and the eighth the shallowest, 1001 ft. In each of these wells the churn drill was used to prepare the way to the top of the salt and the core drill from that point to the bottom. The depth to the top of the salt ranged from 437 ft. in the sixth well to 1283 ft. in the tenth well. The footage cored ranged from 533 ft. in the eighth well to 1168 ft. in the fourth well. Recovery was 100 per cent. in all but the eighth, in which it was 98.7. This is surely a remarkable record.

Private Exploration Stimulated by Government and State Activity

The discovery of potash in oil-well cuttings and the legitimate publicity given thereto by Government and state agencies gave opportunity to some to foster land-leasing and stock-promotion schemes which flourished for a time and led to at least one prosecution for fraudulent use of the mails. Three attempts were actually made to core the salts. One was in southwest Reagan County and another near Odessa in Ector County. Through financial and other difficulties, both these tests were stopped before the salts were reached. The third core test was made in Loving County near the Means well and this penetrated the upper salts.

The publicity attendant on the proposal and final enactment of the potash exploration law by Congress stimulated various oil and other companies with holdings in that field to test their lands. Some of these companies allowed no public use of their results. Others have cooperated fully with Government or state agencies.

American Potash Co. (New Mexico).—In 1926 the Snowden McSweeney interests undertook core drilling on the McNutt permit in sec. 4, T. 21 S., R. 30 E., Eddy County, New Mexico, near the site of one of their oil tests, cuttings from which had yielded the first recognized evidence of the presence of sylvite in the Permian salt basin.[5] The cores were studied in the laboratories of the Geological Survey and the results as announced in December, 1926, were so encouraging that the American Potash Co. was organized to undertake systematic core testing of a considerable tract in that region. Up to January, 1929, this company had made 12 core tests on Government land and two on state land within a radius of a few miles of the first test, all in Eddy County. Eleven of the cores from Government land have been studied by the Geological Survey for their economic content.

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[5] V. H. McNutt: Potash May Make Dry Hole Profitable. *Oil & Gas Jnl.* (1927) **43**, 107, 111, 115, 118.

Gypsy Oil Co.—Another organization that has interested itself in potash exploration in New Mexico is the Gypsy Oil Co., which has made four core tests on state land in Eddy County. This material has also been turned over to the Geological Survey and has been available for examination with the cores of the American Potash Company.

American Potash Co. (Kansas).—A potash core test was drilled near Pratt, Kans., in 1928 by an independent concern called the American Potash Co. and the core was submitted to the U. S. Geological Survey for examination.

American Potash Co. (Texas).—Originally known as the Standard Potash Co., this concern reorganized in April, 1928, under the name American Potash Co., but it has no connection with either of the other companies having the same name nor should it be confused with the American Potash and Chemical Co., which is producing potash from the brines of Searles Lake in California. The American Potash Co. has completed two core tests about 2½ miles apart in southwest Midland County, Texas. The cores have not been available to the U. S. Geological Survey but have been studied and described by Sellards and Schoch,[6] of the Texas Bureau of Economic Geology.

RESULTS OF TESTS

Geologic Results

As a direct or indirect result of Government and state activities in potash exploration, 31 core tests have been made in the Permian salt basin to date (January, 1929), and three others will probably be completed by June 30, the end of the governmental fiscal year, or shortly thereafter. Of those already drilled 20 are in New Mexico, 10 in Texas, of which seven are Government tests, and one in Kansas. Obviously geologic data of the greatest interest have been obtained. The two core tests in Midland County, Texas, made by the former Standard Potash Co., have been studied in considerable detail by Sellards and Schoch, in the paper cited. The other 29 cores are being studied by members of the Geological Survey. A paper by Schaller and Henderson,[7] on the mineralogy of the cores has been prepared for publication. Some of the secondary changes in the salts noted during these studies have already been mentioned.

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[6] E. H. Sellards and E. P. Schoch: *Op. cit.*

[7] W. T. Schaller and E. P. Henderson: The Mineralogy of the Drill Cores Taken in Potash Drilling in Texas and New Mexico. U. S. Geol Survey *Bull.* (in Press).

Economic Results, New Mexico

The first two Government wells drilled in New Mexico encountered sylvite, carnallite and other soluble minerals in beds that were relatively rich but too thin to be of commercial interest. No beds of this sort were recognized in the third well but recovery of core in the upper part of the salt beds in this well was not so good as in the other wells drilled and some beds containing such minerals may possibly have been missed. In this test, however, which was drilled in the SW. ¼ sec. 34, T. 22 S., R. 30 E., Eddy County, New Mexico, a bed of polyhalite 8 ft. 10 in. thick and averaging 11.08 per cent. of potash (K_2O) in the sample as received, equivalent to 18.25 per cent. in the soluble salts, was found at a depth of about 1466 ft. If other associated beds are considered, this interval includes 15½ ft. of beds containing 8.84 per cent. of K_2O in the sample, equivalent to 13.39 per cent. in the soluble salts. This locality is about 28 miles southeast of Carlsbad.

The American Potash Co. and Gypsy Oil Co. have drilled 18 core tests in Eddy County, New Mexico, in an area in which excellent core recoveries have shown beds of sylvite and of polyhalite ranging in thickness from 2 to 7 ft. and carrying from 10 to 20 per cent. of K_2O in the samples as received. A number of other soluble minerals associated with the German potash beds have also been recognized.

Results in Texas

In the interval between the depths of 1935 ft. 7 in. and 1945 ft. in the fourth test drilled in southwestern Ector County, Texas, are two polyhalite beds that have a combined thickness of 6 ft. 7 in. and together contain 12.86 per cent. of K_2O in the sample as received, equivalent to 18.98 per cent. in the soluble salts. These beds are separated by a third bed of halite 2 ft. 10 in. thick. This locality is near the station of Metz on the Texas and Pacific Railway. In the seventh test drilled in southwestern Upton County, about 10 miles north of McCamey on the Orient Railroad, a group of beds 4 ft. 8 in. thick at a depth of 689 ft. 7 in. contains 3 ft. and 7 in. of polyhalite associated with other salts and a 13-in. bed of anhydrite and halite 8 in. above the bottom, which must be discarded. The polyhalite beds contain 9.50 per cent. K_2O in the sample as received, equivalent to 14.73 per cent. in the soluble salts. Another bed of polyhalite at a depth of 848 ft. in the same well is 3 ft. 3 in. thick and contains 11.65 per cent. K_2O in the sample as received, equivalent to 15.06 per cent. in the soluble salts. In the eighth test in northwestern Crockett County, at a depth of 521 ft. there is a 5-ft. bed of polyhalite containing 6.35 per cent. of potash in the sample as received, equivalent to 7.14 per cent. in the soluble salts.

The Government wells drilled in Texas have thus far encountered no sylvite or carnallite or other potash-bearing salts save polyhalite. According to Sellards and Schoch,[8] a 2-ft. layer of soluble salts was penetrated between depths of 1975 and 1985 ft. in the second of the two wells drilled in southwest Midland County by the former Standard Potash Co. Polyhalite occurs in both these wells in various amounts and at different depths below 1900 feet.

Summary of Economic Results

The principal economic data relating to the 10 Government test wells already drilled and reported on are given in Table 1.

In at least two of the 10 localities drilled under Government auspices, beds of polyhalite thick enough and rich enough to suggest commercial exploitation have been discovered and two other localities may be considered as having some commercial possibilities.

Private companies in southeastern New Mexico have prospected in some detail a considerable area underlain by the more soluble potash salts. This area, of which the boundaries have not yet been defined, is shown in Fig. 1. Soluble potash salts have been reported in southwestern Midland County, Texas, but those that have been definitely recognized in that locality are as yet known from only one well.

OUTLOOK

Sufficient supplies of potash salts have been demonstrated to be present in the Permian so that if production problems were solved the United States could feel assured freedom from foreign monopoly during any future national emergency, as in 1910 when existing American potash contracts were voided by German legislation and again in 1915. Since, however, production from such sources can not spring up overnight on sudden demand, it is highly desirable from the viewpoint of the national welfare that the problems of potash production in this field be solved without undue delay, and that commercial exploitation be started if practicable. Problems of transportation and of marketing appear to provide the greatest obstacles at present, though there are problems in mining and chemical engineering that will also require attention. The U. S. Bureau of Mines and the New Mexico Agricultural Experiment Station are now attacking from different angles the problems of the beneficiation or utilization of polyhalite and have already obtained promising results. Some of the private companies are giving serious study to production problems with a view to possible shafting operations. Meanwhile, to gain a better perspective of potash occurrences within the

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[8] E. H. Sellards and E. P. Schoch: *Op. cit.*

TABLE 1.—*Summary of Government Core Tests in the Permian Basin*

Number of Test	Location	Total Depth, Feet	Depth to Top of Salts, Feet	Depth of Better Potash Beds, Feet	Thickness of Better Potash Beds		K_2O in Sample, Per Cent.	K_2O in Soluble Salts, Per Cent.	Remarks
					Feet	Inches			
1	Eddy County, New Mex., NW. ¼ sec. 13, T. 17 S., R. 31 E.	1847	760	1647	4	10	7.58	14.17	Polyhalite and anhydrite. Well discloses more than 30 potash horizons including numerous thin layers of carnallite.
2	Eddy County, New Mex., NW. ¼, SW. ¼ sec. 14, T. 20 S., R. 29 E.	1105	335	809	2	5	13.94	21.25	Polyhalite with anhydrite, halite, etc. Well penetrates more than 20 potash horizons, including some sylvite.
				485	2	8	10.50	17.22	
3	Eddy County, New Mex., SW. ¼, SW. ¼ sec. 34, T. 22 S., R. 30 E.	1505	365	1467	8	10	11.08	18.25	Polyhalite, halite, and anhydrite.
				1459–1475	15	6	8.84	13.39	Polyhalite, halite, and anhydrite and leaner material.
				868	3	2	13.50	20.20	Polyhalite and halite.
				1012	3	7	12.86	21.29	White polyhalite, anhydrite.
				1333	3	3	12.06	18.02	Polyhalite and halite.
									Well penetrates more than 20 potash horizons.
4	Southwest Ector County, Tex., near Metz. on T. & P. Ry., Connell Ranch.	2098	930	1935	6	7	12.86	18.98	Polyhalite and halite in two beds separated by about 3 ft. of halite.
				1280	3	7	9.53	15.85	Polyhalite and anhydrite.
				1303	3	9	9.32	15.68	Polyhalite and anhydrite.
				1309	3	3	10.30	13.31	Polyhalite and halite.
									Well penetrates more than 10 potash horizons.
5	Crockett County, Tex., Harris Bros. Ranch, NW. ¼ sec. 16, G. C. & S. F. Ry. Survey.	1799	1161	1370	5	8	5.84	6.87	Polyhalite, halite, and anhydrite. Excludes a 10-in. layer of halite and anhydrite at 1375 ft.
				1682	1	0	13.52	19.78	Richest bed—polyhalite.
				1210	1	8	7.67	9.34	Polyhalite with halite.
6	Upton County, Tex., Sec. 100, Burleson-Sun lease, about 3 miles north of McCamey.	1501	437	1168	1	8	10.00	18.20	Polyhalite and anhydrite—richest bed.
				632	2	3	8.24	9.04	Polyhalite and halite.
				902	1	11	9.22	11.50	Polyhalite, halite, anhydrite.
				1213	1	11	8.87	11.11	Polyhalite, halite, anhydrite.
7	Upton County, Tex., sec. 4, William Teer Survey, Hughes Roxana lease, 10 miles north of McCamey.	1230	445	690	3	7	9.50	14.73	Excludes 8 in. of halite and anhydrite.
				848	3	3	11.65	15.06	Polyhalite, etc.
									Tests 6 and 7 both penetrated about 10 potash horizons.
8	Crockett County, Tex., sec. 5, block 14, Univ. land.	1001	468	521	5	2	6.35	7.14	Polyhalite, anhydrite and halite. About 10 potash horizons penetrated.
				529	0	5	11.22	14.10	Richest bed.
9	Reagan County, Tex., NW. ¼, SW. ¼ sec. 1, Block 2, Univ. land, Big Lake district.	1700	1104	1526	2	11	8.27	10.94	Polyhalite and halite.
				1310	2	3	7.62	9.99	About 15 potash horizons penetrated.
10	Glasscock County, Tex., NE. ¼ sec. 14, Block 35, T. 2 S. T. & P. Survey, Houston Ranch.	1892	1283	1370	3	2	7.90	10.75	Polyhalite and halite. Excludes 13 in. waste.
				1575	0	11	15.12	24.20	Richest bed—pure polyhalite. More than 10 potash horizons penetrated.

Permian salt basin, the Government should continue its exploration at least through the remaining two years of the five-year period authorized by Congress. The U. S. Geological Survey, by its study of well cuttings prior to the passage of the potash act, had already reduced the area potentially subject to core testing to about one-sixth of its original size. The remaining territory, however, is still so vast that the relatively few tests possible under the authorization will give at best a very imperfect picture of this great potash field.

DISCUSSION

MEMBER.—What is the length of core section pulled?

W. B. LANG.—A 10-ft. core barrel is usually used and unless difficulties have been experienced or sensed, the barrel is filled before the rods are pulled.

MEMBER.—Are the polyhalite and sylvite beds so closely associated that they can be worked together?

W. B. LANG.—In the majority of cases the polyhalite and sylvite beds are separated by such thicknesses of salt that is not potash bearing as to make them independent mining units although penetrated by the same shaft.

MEMBER.—What is the position of the potash beds in the salt column?

W. B. LANG.—The potash-bearing beds are contained in the Upper Salt Series.

MEMBER.—Have any holes been drilled since the paper was written?

G. R. MANSFIELD.—One. The Government has drilled 11 wells. The five-year program should complete about 20 holes.

The Barite Industry in Missouri

W. M. WEIGEL,* ST. LOUIS, MO.

(New York Meeting, February, 1929)

ECONOMIC deposits of barite occur in Missouri in two main districts. The most important, the Southeastern or Washington County district, is in the southeastern part of the state, mostly in Washington County with a small extension into the northeastern part of St. Francois County. Roughly, it embraces nine townships or about 200,000 acres and accounts for from 80 to 90 per cent. of the state production. The Central district, in the central part of the state, includes most of Morgan and Miller counties, and parts of Camden, Benton, Moniteau and Cole counties. The area embraced is much larger than the Washington County district, but the deposits are more infrequent. The location and general area covered by the two districts is shown in Fig. 1. There is another small area in southern Franklin County but so far its production has been of small consequence. It might be considered as a northward extension of the Washington County field. An area in the central part of Hickory County has also produced a small amount. The outline of the districts as shown indicates the general area in which deposits have been found. The total area covered by deposits of barite is only a small part of this, as, in the Central district especially, there may be miles of supposedly barren areas between ore-bearing areas of an acre or two in extent.

The date when barite was first mined in Missouri cannot be definitely stated, but the industry probably dates back to about 1860. The occurrence of barite was known long before this, as it was encountered in the lead mines in Washington County and mentioned by several earlier writers.[1] Missouri is reported as producing 8000 tons in 1882 so it is evident the industry was well under way in the seventies in order to have arrived at this substantial production. By 1893 Missouri production had increased so that it ranked with that of Virginia, then the largest producer. It increased until in 1914 Missouri accounted for about 65 per cent. of the total United States output.

RECENT PRODUCTION OF BARITE

Table 1, compiled from *Mineral Resources*, published by the United States Geological Survey and Bureau of Mines, gives the production of

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* Mineral Technologist, Missouri Pacific Railroad.

[1] A. Litton: Missouri Geol. Survey (1855) 12.

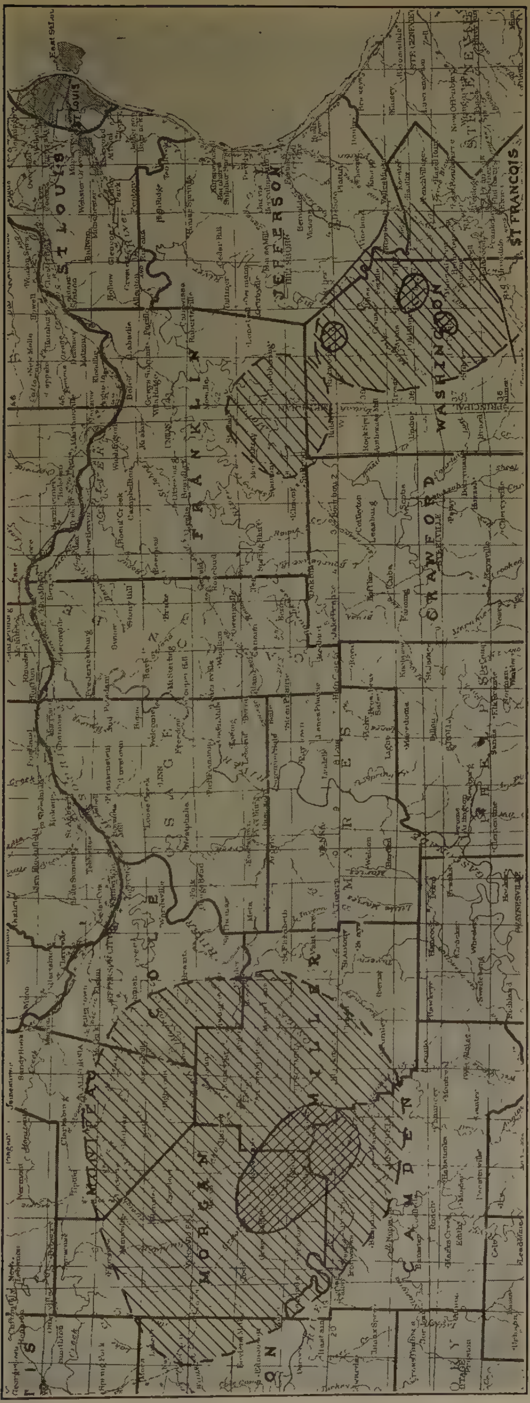

Fig. 1.—Barite districts in Missouri.
Areas containing known deposits are indicated by diagonal lines; areas of greatest development, by crosshatching.

crude barite in Missouri and the total in the United States for the past 15 years. Georgia became an important producer about 1914 and production for that state is included beginning with 1915.

TABLE 1.—*Crude Barites Produced or Marketed*

Year	Missouri, Tons[a]	Georgia, Tons	Total United States, Tons
1913	31,131		45,298
1914	33,317		52,747
1915	39,113	31,027	108,547
1916	58,223	104,784	221,952
1917	59,046	111,300	206,888
1918	49,094	69,318	155,368
1919	73,247	85,303	209,330
1920	99,654	84,644	228,113
1921	25,200	24,614	66,369
1922	66,421	57,045	155,040
1923	81,701	83,291	214,183
1924	77,189	71,776	196,332
1925	101,056	65,936	228,063
1926	118,919	77,654	237,875
1927	111,456	94,039	254,265

[a] All tons = 2000 lb.

During the past 15 years production in Missouri has increased 380 per cent. and in the United States 550 per cent. This rate of increase is probably more than for any other similar mineral commodity and speaks well for the future of the industry. It is accounted for by the increased use of barite in manufacturing and the arts and the development of new uses.

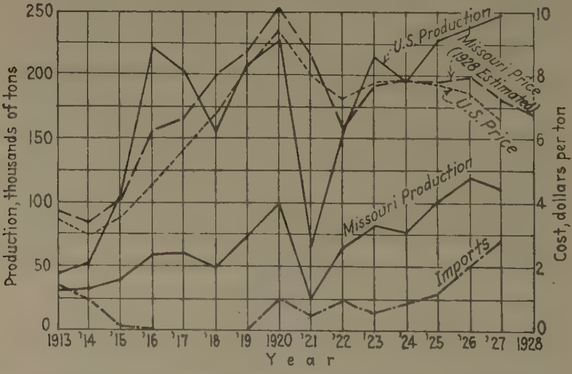

FIG. 2.—BARITE PRODUCTION, PRICES AND IMPORTS, UNITED STATES AND MISSOURI.

Fig. 2 shows graphically the total production in the United States and that in Missouri, and the average yearly prices, since 1913.

ECONOMIC CONDITION OF BARITE INDUSTRY IN MISSOURI

Since the depression of 1921 the barite industry in Missouri has been quite stable and fairly prosperous. Production has increased steadily and during the past three years more modern methods have been introduced for a considerable proportion of the total production, yielding increased recoveries and reduction of costs. Some of the larger producers are conducting systematic research to extend the uses of barite and are planning their operations with a vision for the future. Prices have been fairly steady up to the summer of 1928, when there was a drop of $0.75 to $1.00 per ton for crude.

This was probably due to a lessening in demand with but slight decrease in production to counteract it, and to the increase in imports of which the Missouri market was only then beginning to feel the effect. From 1915 to 1919, there was no importation of European barite. From 1919 to 1924, imports were not a serious factor in the American market, but since then they have risen steadily until in 1927 they equaled 28 per cent. of the domestic production. Imports are absorbed largely along the Atlantic seaboard, and here they come into direct competition with the Georgia and Tennessee production. This pressure caused the southeastern producers to seek new outlets in the upper Mississippi Valley and lake markets, which brought them into competition with the Missouri production. The consequent oversupply for these markets caused a drop in price (Fig. 2). Even with decreased prices there has been little curtailment of production in Missouri and the producers are optimistic as to the future.

GEOLOGY OF WASHINGTON COUNTY DISTRICT

Nearly all the formations of the Washington County district are of Cambrian age. There is only a small area of Ordivician (Roubidoux) in the southwestern part of the district. With the exception of small areas in the creek valleys in the northeastern part, eastern part and the southeastern margin, the Potosi is the lowest formation exposed. It forms the surface of a large part of the area.

Extending north from the town of Potosi for 10 miles, the Gasconade formation covers the top of a long narrow ridge while the Emminence is exposed on the sides of this elevation between the Gasconade and underlying Potosi. The Gasconade is the top formation exposed in a large part of the northwestern and southwestern areas, but generally beyond the productive areas. The Gasconade and Emminence also cap some of the individual hills and high points of the main productive portion of the district.

The formations of the Washington County district and also those in the Central District, according to the Missouri Geological Survey, are:

- Lower Ordivician
  - Jefferson City
  - Roubidoux
- Cambrian
  - Gasconade
  - Proctor
  - Emminence
  - Potosi
  - Derby-Doe Run
  - Davis
  - Bonne Terre
  - Lamotte sandstone

The Davis is the oldest exposed in the eastern creek valleys, partly on account of cutting down by the streams, but more because of faulting and folding in these areas which resulted in structural changes that brought the lower formations up within reach of erosive action. Dolomites constitute by far the greatest part of the Cambrian formations, although some contain shale, chert and sandstone. The Davis especially contains much shale, shaly and sandy dolomite, and conglomerate, and often weathers platy.

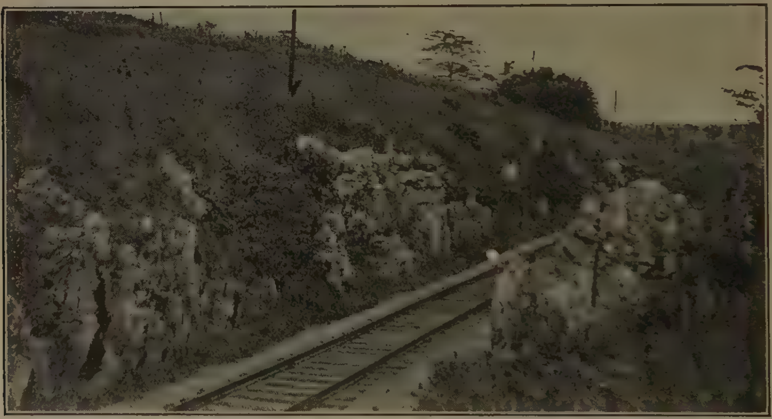

Fig. 3.—Potosi dolomite covered with barite-bearing clay, near Cadet, Mo.

No economic deposits of barite have been found below the Potosi. A few occurrences are known where the deposit rests on the Derby-Doe Run but it has no genetic connection with the latter. The Potosi formation, the most important barite producer, is entirely dolomite and is characterized by its large content of drusy quartz, chalcedony and chert. This gives the rock a distinctive, rough, uneven appearance on weathered surfaces. Fig. 3 shows an exposed top surface of Potosi under its barite-carrying clay mantle and illustrates the uneven weathering. Large masses of the quartz druses often project several inches from the weathered surface where the less resistant dolomite has weathered from around them. In small areas these druses may be almost entirely absent, but the rock may be identified by its distinctive granular structure. The drusy forms occur generally as irregular masses more or less connected into pipes. Some of the smaller ones resemble geodes. They are inclined to occur most abundantly in certain horizons. The Potosi formation[2] is from 250 to 300 ft. thick in the eastern part of the district but thins to the northwest, where it probably does not exceed 100 to 150 ft. Most of the producing barite deposits are in the Potosi or in the clay mantle resting directly upon it. When the quartz and chert are absent the dolomite may be quite low in impurities. A sample taken near Cadet gave the following partial analysis: CaO, 30.7 per cent.; MgO, 18.8 per cent.; SiO, 0.9 per cent.; R_2O_3, 0.2 per cent.

Limiting the district on the northeast is a large fault or series of faults having a general northwest-southeast strike. The downthrow is on the northeast side. About two miles southeast of Hopewell there is another major fault, which strikes northeast and southwest, or approximately at right angles to the other, with the downthrow on the northwest side. The lines of strike of these two faults intersect 7 to 8 miles east of the Washington-St. Francois County line. The barite-producing area falls within the angle between these two major faults, as barite is not found north or south of them. Their approximate location is shown on Fig. 1.

The Emminence, the Proctor of Tarr,[3] is of some importance as a barite producer but much less so than the Potosi. It occurs principally in the western half of the district and along the ridge northward from Potosi. It varies from 80 to 120 ft. in thickness, but is rarely exposed in bluffs or escarpments, as the outcrops are broad and flattened out as a result of weathering or are covered with a mantle of clay and chert from the Gasconade. The Emminence in this district is nearly pure dolomite and does not contain the numerous quartz druses found in the Potosi. Chert is found at intervals but is infrequent. According to the most recent work of Prof. C. L. Dake for the Missouri Geological Survey,[4] the Proctor is missing in this part of the Washington County district, the Gasconade resting directly upon the Emminence.

The Gasconade, covering some of the area in the southwestern, middle western and northwestern parts of the area, consists of dolomites, sandstones, cherts, and occasionally shale. It does not carry any of the barite deposits in the Washington County district, although it is the principal ore-bearing formation in the Central district. The small areas of Roubidoux are also of no economic importance in this area. When present it is mostly sandstone.

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[2] W A. Tarr: The Barite Deposits of Missouri. Univ. of Missouri *Studies* (1918) **111**, 28.

[3] W. A. Tarr: *Loc. cit.*, 30.

[4] Private communication to the author.

Central District Geology

The geological section given for the Washington County district applies also to the Central district. In addition there are a few very small areas of the Mississippian and Pennsylvanian series, but they are of no economic importance as far as barite is concerned.

The lowest formation exposed is the Proctor, in the valley of the Osage River, where local rolls have elevated it sufficiently to be cut into by the river gradient.

The Gasconade forms the surface of much of the southern half of the district in the main Osage River valley and its tributaries, while the Roubidoux covers the ridges and high land of the southern part and is about equal in area to the Gasconade. The northern part of the area is covered almost entirely with the Jefferson City.

The Gasconade formation in this area is from 240 to 290 ft. thick, composed mainly of alternating massive and thin beds of crystalline gray dolomite. Some of the dolomite is cherty. There are also beds of chert and sometimes sandstone, which usually occurs as lenses. The chert in the dolomite is in nodules, small masses or thin beds. Some of the more prominent beds of chert are several feet thick. The basal member of the Gasconade is the Gunter sandstone, quite variable in thickness, ranging from 5 to 35 ft. It has no economic importance as a barite producer.

The Roubidoux is made up of an irregular series of dolomite, cherty dolomite, sandstone and chert beds. Few of them are persistent. The total thickness varies from 70 to 160 ft. Most of the dolomite is fine grained, varying from gray to buff. The chert beds may attain a thickness of 30 ft. in places and are more frequent in the dolomite beds of the Roubidoux than in those of the Gasconade. The sandstone beds are seldom more than 2 ft. thick.

The Gasconade contains barite deposits; the Roubidoux, less than the Gasconade; and the Jefferson City only a small amount. Most of the deposits are found on top of the Gasconade or near the contact of the Gasconade and Roubidoux.

The formations of the district have a gentle dip to the northwest. There are numerous local folds, or rather flexures, but not of enough magnitude to greatly disturb the general structure of the area. Faulting is not common. Faults are usually short and of small displacement. Southwest of Linn Creek, however, faulting is more pronounced.

Economic Geology of Barite, Washington County District

In this district the barite occurs in three ways: (1) in place in the rock, as veins, disseminated as small crystals or clusters; (2) in solution cavities of large or small size; (3) as residual deposits, which are economically the most important.

Veins

At various points in the district where the Potosi has been exposed by stream action or in bluffs, veins of barite may be observed. Very little attempt has been made to develop these; the cost of extraction would be much greater than from the residual deposits, as the veins are usually so narrow, not over a few inches wide, that much barren rock would have to be removed in their exploitation. These veins sometimes occur as networks with connecting stringers a fraction of an inch thick. It is evident that much of the barite in the residual deposits was derived from the decay and weathering of rock containing barite veins, but also that some of it came from solution cavities and individual clusters. It is not unusual to find fragments of solid barite of roughly boulder form, weighing from 100 to 200 lb., which evidently were not originally deposited in vein form. Modifications of the vein deposits are the fractured areas sometimes found where the fissures between the rock fragments are filled with barite, quartz and sometimes chert. Undoubtedly some of the rich residual deposits of small extent are directly under the original location of fractured or fissured areas in the pre-existing parent rock. These areas allowed rapid weathering and decay by surface agencies and the more resistant barite has remained practically without horizontal displacement in the residual clay.

Barite may be found disseminated in the rock in small grains, but more often in irregular masses, 4 to 10 in. dia. These usually occur near a fissured area; seldom in the massive beds of rock where mineralization has not taken place. In addition to the barite, clusters and coatings of drusy quartz fill fissures and small cavities in the rock.

Solution Cavities

Solution cavities, or small caves, are sometimes encountered which are partly filled with sandstone and other rock fragments but contain barite, occasionally in pieces weighing 200 to 300 lb., and sometimes other minerals such as calcite or galena. These are more common in the Gasconade formation of the Central district than in Washington County. While interesting, and occasionally good producers for a short time, this type of deposit has not produced great amounts of barite.

It is entirely probable that some of the deposits in the Central district now called residual were originally of this type and have been reduced to their present state by decay and caving of the overlying rocks. In two deposits, one near Bagnell and the other south of the Osage River, southwest of Bagnell, deposits have been worked in which a rim of solid rock remains in place on the hillside below the deposit, with its top considerably higher than the bottom of the residual clay in the pocket or cavity. A generalized section of one of these is shown in Fig. 4.

Residual Deposits

Residual deposits are by far the most important in both districts, as they are more easily worked and the barite is more concentrated in this form of deposit.

This type contains the barite as loose fragments embedded and scattered through clay. More or less chert, quartz and larger boulders of dolomite and sandstone are mixed with the clay. The barite fragments vary in size from fines or slimes to masses weighing 200 lb. or more. Generally the larger masses are near the bottom of the deposit, sometimes on bed rock with fragments of this or overlying strata, but this is not always the case. The clay is very fine, plastic and almost free of grit and varies from light to very dark red. A deep brownish red is the most common color.

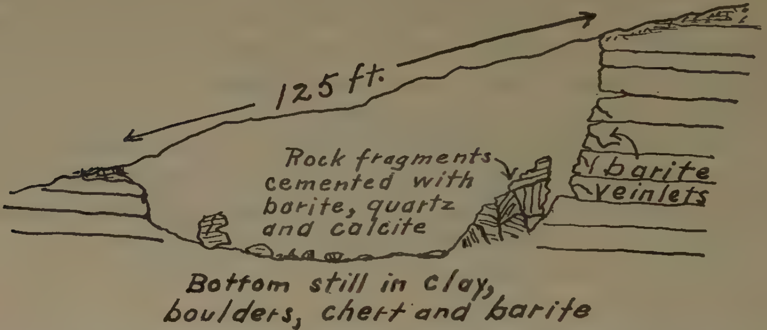
125 ft.
Rock fragments cemented with barite, quartz and calcite
barite veinlets
Bottom still in clay, boulders, chert and barite

FIG. 4.—TYPICAL CROSS-SECTION OF ONE FORM OF CENTRAL DISTRICT DEPOSIT PARTLY MINED OUT.

Most of the barite is pure white on a fresh fracture, although always coated on the outside with a film of limonite or iron oxide, which is so thin that it has little effect on the chemical analysis. The rough lumps are made up of tabular crystals and are quite friable, as sharp edges can be crumbled and broken off between the fingers. Many of the larger unbroken lumps have deep cavities on their faces, lined with a layer of limonite 1⁄32 to 1⁄16 in. thick. This adheres closely to the barite and cannot be removed by washing. It is the source of most of the iron content of these ores.

In the Washington County district residual deposits occur in the clay mantle over the top and slopes of the hills. Sometimes a continuous area of over 100 acres will be ore-bearing, with a thickness varying from 4 to 30 ft. Barite is seldom found in the valleys; the richest areas are usually on the gentle slopes part way down the hill; the tops of the hills are inclined to be lean. In some areas the barite may be quite uniformly distributed in the clay, but more often it occurs in concentrated layers with lean beds of clay intervening; it is usually more concentrated in leads or runs, which may be 10 to 20 ft. wide and several hundred feet long. They have no definite relation with the slope of the hill.

One form of concentration in layers shows the following section, which is typical of an area near Cadet:

Surface soil	1 ft. to 1 ft. 6 in.
Chert, gravel and clay	1 ft. 6 in. to 2 ft.
Clay, barite, chert and quartz, barite in small fragments, not over 2 in.	2 ft. 6 in. to 4 ft.
Practically barren dark red clay	3 ft. to 4 ft.
Red clay, large chert, and dolomite boulders and large masses of barite, sometimes in almost continuous sheets 4 to 6 in. thick and several square feet horizontal extent	1 ft. to 2 ft. 6 in.
Red clay, practically barren, depth to bed rock not exposed.	

The barite content of the workable deposits varies considerably. In small areas and pockets it may run as high as 20 to 25 per cent. Large areas being mined by steam shovel will probably average under 10 per cent., and considerable areas have been mined which probably yielded less than 5 per cent.

Economic Geology, Central District

The deposits of the Central district are similar to those in Washington County in that the barite occurs scattered through residual beds or pockets of red clay resulting from the decay and weathering of the overlying rocks. They differ in that the individual deposits are much smaller in horizontal dimensions, although often considerably deeper. Very few deposits have a horizontal dimension greater than 200 ft. This is probably because in this district the hillsides on which the deposits are found are steeper and the topography in general more rugged. This is true of the deposits found in the Gasconade formation. Those noted in the Jefferson City were apparently quite shallow and smaller in area. As mentioned, a rim or ridge of rock is often found on the slope below the orebody. This may indicate that the original deposition occurred in a cave or in a broken area, which allowed free ingress of the barite-laden solutions. Rim rock is always found on the upper side of the deposit on a slope, and usually at the ends. In all cases the longer axis of the pocket lies parallel to the crest of the hill.

Some of the deposits have been opened to a depth of 50 ft. below the surface, but 15 to 20 ft. is more common. The barite content is higher than in the Washington County district. Many deposits contain 20 per cent. or more barite and some old dumps were observed that would contain 12 to 15 per cent. barite, from which the larger pieces, 2 in. and over, had been picked by hand.

The barite occurs in smaller fragments than in Washington County; in some deposits there is a large percentage of "glass tiff," a local name for the transparent crystalline form of barite. The iron content is lower and the larger pieces do not have the limonite-lined cavities characteristic of much of the barite occurring in large-lump form in the Washington County District. The associated clay is much the same in texture and appearance. Drusy quartz is lacking but there is considerable chert, chert gravel and fragments of hard dolomite, sandstone and some calcite.

In the Washington County ore, the barite is practically free of any adhering rock, while in the Central district a considerable amount, especially in the smaller sizes, around 1 in., is strongly attached to rock fragments of about the same size.

In the bottoms of the orebodies large angular boulders of sandstone or dolomite are often found fairly close together, with the crevices between adjacent boulders filled with stiff red clay and tabular crystals of barite conforming more or less to the shape of the opening.

In this district two or three deposits may often be found along the slopes in one short valley with no other known for a distance of two or three miles. Hence, while the economic deposits are scattered over a very large area, the actual area of the known barite-bearing ground is small as compared to the Washington County district. Much of the area is rather inaccessible at present, and has not been as systematically prospected as has Washington County so that there is reason to believe that many new deposits will be found by careful and intelligent search.

ORIGIN OF THE BARITE DEPOSITS

There is no question as to the source of the barite in the residual deposits. They are the result of the accumulation and concentration of the barite originally deposited in veins and cavities of the overlying strata, released by solution and weathering of the more soluble rock minerals, mainly calcium and magnesium carbonates, of which these rocks were composed. The clays are also the result of this action, making conditions proper for the formation of the orebodies when not subjected to erosive action. This applies to both districts.

The original source of the barium is not so easily determined and is the subject of considerable controversy. One school advances the theory that the barite in the veins and cavities in the sedimentaries was precipitated, or crystallized from hot ascending magmatic solutions originating in the magma that later crystallized and now forms the igneous floor of the Ozark Uplift; the faults and fractures due to folding and flexure afforded passage for the mineralizing solutions. There are many supporters of this theory and some of the evidence advanced in support of it is good.

The other school believes that the barite and associated minerals were deposited by descending waters; this theory is coming to be accepted by many geologists who have spent much time in a study of the Ozark ore deposits, the barite as well as the lead and zinc. In support of this there is no evidence of the effect of heated solutions in the vein deposits or in any of the other types. No barite has been found in the formations below the Potosi and it is argued that if the barium were carried in ascending waters, deposition would have occurred at least in places in the several hundred feet of sedimentaries between the igneous floor and the Potosi.

The wide distribution of the deposits over considerable areas do not agree apparently with the theory of magmatic origin, as deposits from this source usually have a different form and bear the earmarks of the action of the transporting medium. Massive vein or "bull" quartz is entirely absent and it is known that the chert, chalcedony, and drusy quartz, contemporary minerals with the barite, are derived from or due to the action of descending waters. Galena is often found associated with the barite, but in every instance the barite was deposited after the galena; it is quite common to find galena crystals inclosed in barite, but never the reverse.

Barite Ore Reserves

Even an approximate estimate of the tonnage of barite still recoverable in both the districts is difficult or impossible. The total Missouri production to date is about 1,750,000 tons; taking into account improved methods of mining and treatment, which permit more complete recovery and the working over of areas already gone over by hand methods, the indications are that future production will greatly exceed the past from the present known ore-bearing areas. Also new areas are being found and the limits of the fields extended.

Mining Methods

Prospecting

Prospecting methods are comparatively simple. The general lay or topography of favorable ground is known. When such areas are found an attempt is made to locate rock exposures, to determine whether the formation is favorable. In the Washington County district, the Potosi, or possibly the Emminence, is sought; in the Central district, the Gasconade and Roubidoux, especially the contact between them. In cultivated fields "float" barite may be found; on timbered slopes surface barite is rare, although it may be encountered within a foot or two of the surface.

When promising ground is found, prospecting is continued by digging test pits. If barite is found other pits are sunk in the immediate vicinity, and in the Central district this operation is extended to the limits of the deposit. In the Washington County district an attempt is made to define the "leads" or "runs" of rich ground. One method of prospecting is to drive a pointed steel rod into the ground to the depth required and carefully withdraw it. If barite is struck the point of the rod will be whitened by the soft mineral. Boulders of chert or hard rock will not whiten the rod.

In prospecting large areas for shovel mining, test pits are put down at regular intervals and the entire excavated material is carefully sorted in an attempt to determine the grade. The results are always low, as the fine barite, which would be recovered on jigs or tables, cannot all be separated by hand.

Hand Mining

Hand methods of mining still produce well over one-half the total production, and all the miners or "tiff diggers" are familiar with these methods. The miner's tools consist of a pick, shovel, sledge, small hammer with one end finished like a hatchet or chisel, hand windlass, rope and bucket and an ax. A pit or shallow shaft, about 5 ft. dia. and of no particular shape, is started on the selected location. The pit may be circular but if cribbing is used in the upper part it is square. The dirt is thrown out until a depth too great for this is reached. Many pits never exceed this depth. On the surface the dirt is worked over by pick and shovel and the lumps of barite are sorted out. Large pieces are cleaned by hand with the aid of the edge-faced hammer. If a good run of ore is found it is followed by drifting as far as possible before the roof caves. When this occurs, or bed rock is reached, another pit is started, often within 8 or 10 ft. of the first, and in the direction of the best ore. Drifts from one pit often lead into those from an adjacent one. Pits may be anywhere from 4 to 30 ft. deep. When a windlass is necessary, two or three men usually work together, one in the pit and one or two on the surface.

Fig. 5 shows an area near Potosi which has been intensively worked by hand mining. This particular area has been worked over at least three times, illustrating the poor recovery of the total barite in the ground by hand methods. Hand mining methods are most profitable and best adapted to deposits where the barite occurs in large lumps or where the surface of the bed rock is irregular, making mechanical methods difficult.

The operators having a lease on the land, or the owner wishing to work his own property, builds a number of small houses on the place. These are furnished, usually rent-free, to the miners, who work always on piece work or contract for a price per ton of barite cleaned and piled up by the side of the pit ready to load into trucks or wagons. This price varies, partly with the richness of the ground but mostly with the distance from the railroad, from $2.50 to $4.00 per ton. The market price also affects the contract price for digging.

The average miner will put out about 2½ tons per week. He is independent, his wants are few, he never works on Saturday and often not on Monday. On other days he will probably not average over 6 hr. per day of actual labor.

Hand methods differ in the Central district because the deposits are usually on a steeper slope and of less horizontal extent. The deposits are opened by starting a cut into the hillside and widening this out as it deepens. A nearly vertical face is developed and broken down by picks, or sometimes by barring from above. The barite and clay are separated on the pit floor and hauled out separately. Large boulders are left in place and mined around (Fig. 4).

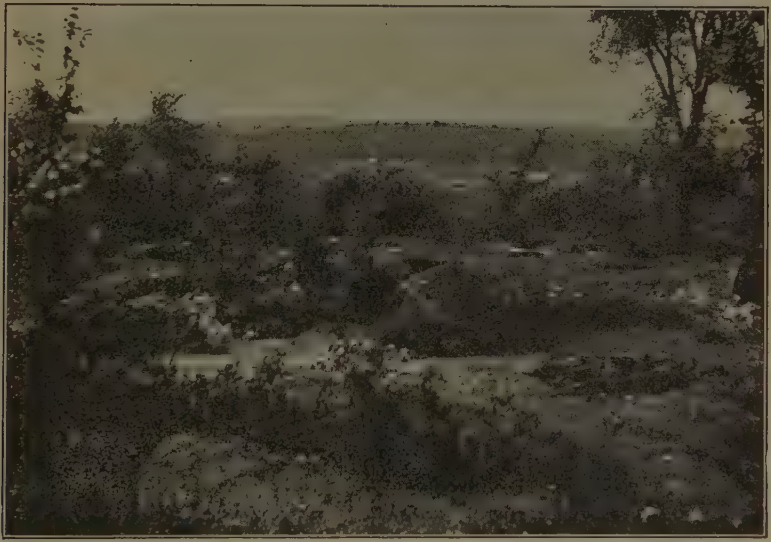

Fig. 5.—Barite ground worked over by hand methods, near Potosi, Mo.

Costs of Hand Mining

Mining is paid for on a contract basis, the average price in 1928 being about $3 per ton.

Royalties paid to the landowner depend on distance from transportation and other factors. They vary from 50 c. per ton up to $2, in rare instances, per ton of barite shipped. The landowner can afford to take a lower royalty from mechanical mining than from hand mining, as the tonnage extracted per acre will be much more and the royalty is more quickly realized.

Mechanical Mining Methods

Attempts to work the Washington County deposits by power shovels followed by mechanical washing and concentrating are only four or five years old. The first operations were those of the National Pigments and Chemical Co. near Cadet and Old mines. This company now has two shovels supplying two washing plants. The Superior Mineral Co. is also operating an electric shovel near Cadet.

The best conditions for power shovels are in gently sloping ground where the deposit covers a large area. Power shovels are especially suitable where the barite occurs in small pieces, usually called "gravel tiff," uniformly distributed through the clay rather than in large lumps or sheets. Much of the ground so far worked has previously been mined by hand, but has had only the larger pieces of barite removed. So far, 12 to 15 ft. is the greatest depth excavated by power shovels. The entire thickness is removed in one cut and no overburden is stripped, everything excavated being sent to the washer. Shovels have from ¾ to 1½-yd. dippers and are operated by steam, gasoline or electric power. All are of the caterpillar type with full revolving motion. As the ground is fairly soft and digs easily, their output is limited rather by the ability to haul away the ore than by the capacity of the shovel. The shovels are served by 5-yd. side-dump cars pulled by 8 or 10-ton gasoline locomotives, delivering to the washing plant with hauls of from 1000 to 2500 ft. Two locomotives, each handling two cars, serve each shovel. The practice at the three operations is much the same and similar to all work of this kind. Blasting the ground ahead of the shovels is not necessary. The shovel of the Superior Mineral Co. is shown in Fig. 6 and the operations at one of the pits of the National Pigment and Chemical Co. are shown in Fig. 7.

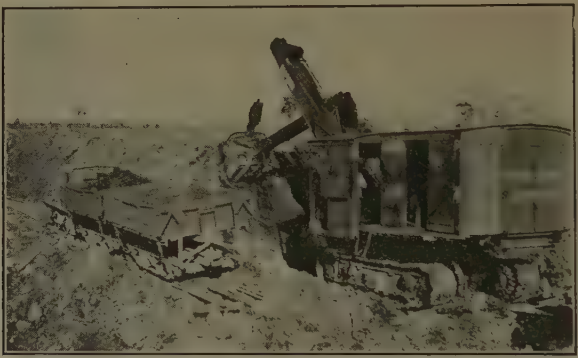

FIG. 6.—MINING BARITE WITH ELECTRIC SHOVEL, NEAR CADET, MO.

On account of its uneven surface, it is impossible to excavate all the "dirt" to the bed rock. This leaves many depressions and hollows unmined, and where these are rich enough they will be worked out by hand methods after the shovel has worked over them. If they contain barite, it is in large boulder form, while that in the overlying clay is mostly the "gravel" variety.

Fig. 7.—Mining barite with gasoline shovel, near Cadet, Mo.

Costs of mining on the basis of barite recovered are not available, but the actual cost of excavation and loading would be about the same as for similar operations elsewhere.

Only one attempt has been made in the Central district to mine with a power shovel. This was of short duration because of financial matters not directly connected with the mining. The nearest approach to mechanical methods is in the working over of some waste dumps by horse-drawn scrapers after the clay is loosened by plowing. In one such operation the cost of excavating and transporting the material an average distance of 150 ft. to the platform above the washer was approximately 12 c. per ton. A team and driver cost 50 c. per hour. Day labor is paid from 30 to 35 c. per hour. Teams could not be secured at this price except during the slack season on near-by farms.

MILLING AND CLEANING

Hand Methods

Over one-half the barite produced is cleaned by hand methods. In the Washington County district the large lumps are cleaned as thoroughly as possible by scraping or cutting off the adhering clay with a hatchet, or hammer with one cutting edge. They are allowed to dry, and much of the remaining clay falls off, as its high shrinkage causes it to crack or loosen during drying and exposure to the weather. The smaller fragments are piled on a platform and allowed to weather and dry, which causes much of the adhering clay to loosen and fall off. The barite is then placed in a rocker and worked back and forth so that the abrasion of the fragments against each other removes most of the remaining coating. A typical rocker is shown in Fig. 8. It consists of a box 12 to 14 in. deep, 18 to 22 in. wide, and 30 to 36 in. long, mounted on a pair of rough rockers. The bottom is made of old wire-screen cloth or sheet iron punched full of holes. The openings in the sheet metal or screen are from ½ to ¾ in. dia. Fine barite that passes through the screen is lost, of course.

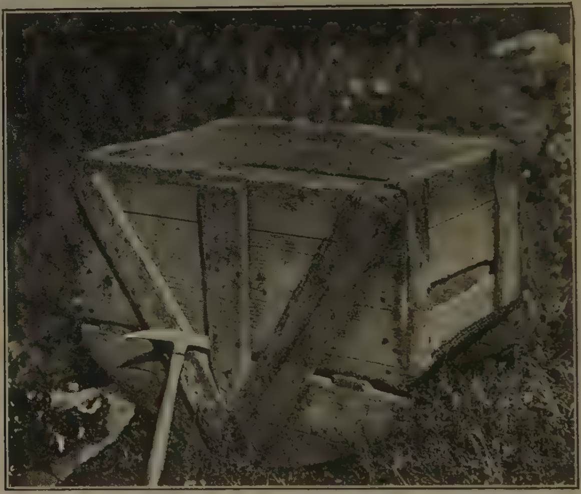

Fig. 8.—Rocker used in Washington County district.

Rockers are seldom used in the Central district. The clay separates more readily, and after weathering for a while, the operation of loading it with a fork into wagons and trucks and unloading at the railroad is usually sufficient to reduce the clay to the permissible limit.

Combined Hand and Mechanical Cleaning

Recently there have been installed in the Washington County District several combined hand and mechanical washers. A washer of this type consists of a small single-log washer 8 to 10 ft. long operated by a gasoline engine. This is fed by hand with ore brought from near-by in barrows or wagons. At one side is a hand jig of the usual type. The barite and waste rock freed of clay by the log are separated on the jig and shoveled on to separate piles. The hutch product is as high grade as the barite taken from the screen.

This outfit can be moved from place to place at little expense and quickly set up and placed in operation wherever water is available. It offers a ready method of reworking areas previously mined by hand, which are not large enough to justify the cost of a large permanent washer and yet contain a sufficient amount of fine barite to be worth saving. A combined log and jig is operated by two men. Some combinations in operation near Potosi and Richwoods are turning out a nice product. One of those at Richwoods has a 12 ft. log and three jigs.

Mechanical Washing or Concentration

There are now three mechanical washers in the Washington County district and three or four in the Central district. Only two of the latter, however, may be considered complete. The reasons for the delay in adopting power-driven washers have been the low cost of labor, the ease with which the ore can be cleaned by hand, and the uncertainty of having deposits of sufficient size to justify the capital outlay.

The concentration problem is comparatively simple. The ore consists of a mixture of clay, boulders and fragments of stone, chert, quartz, gravel, and barite. Occasionally galena is found, but usually it occurs in large pieces and is picked out by hand or caught in the first cell of the jigs.

Where the barite is coated with a hard film or layer of limonite, matters are complicated somewhat, as even if the limonite is crushed fine enough to free it from the barite, its density (3.8) is so near that of barite (4.3 to 4.6) that separation by gravity methods is difficult, especially as the barite tends to break into tabular and the limonite into granular form.

The barite, with the exception of some in the Central district, is free of adhering rock, so crushing to free locked grains is not necessary except in the instances mentioned and where an attempt is made to separate the limonite coating.

Standard practice is the use of log washers to remove the clay, followed by comparatively coarse crushing, jigging and table treatment of the fines. In detail the practice varies from a very simple flow sheet to a rather elaborate one, depending partly on the ideas of the operators as to what constitutes true efficiency, but more on the grade of concentrates necessary to supply the different markets. It must be remembered that barite is a cheap mineral commodity with a value of from $6 to $8 per ton f.o.b. cars, and evidently refinements of concentrating methods to secure a high percentage of recovery cannot be employed unless the cost of these extra refinements is extremely low.

Another factor to be considered is that a very large amount of the waste material, the clay, is eliminated by the log washer in the first operation, therefore if the original feed to the logs contains 7 to 8 per cent. barite the product leaving the logs and going to the jigs may contain from 25 to 45 per cent.

The two washing plants of the National Pigments and Chemical Co. near Cadet are much the same; each handles 400 to 600 tons per 9 hr. Power is furnished by Diesel engines. The ore is brought in on an elevated track in 5-yd. cars and is dumped directly into a wide apron conveyor or feeder. This discharges onto the apex of a wedge-shaped grizzly, the under size passing to the logs and the oversize to platforms with a grizzly floor, where the large barren boulders, roots and other rubbish are thrown out by hand. There are two double logs, each 30 ft. long. The overflow from the logs goes to the mud pond. The washed barite and rock is conveyed to a Symons crusher, reduced to about ¾ in., then sent to bins. From the bins it is fed to three three-cell jigs with beds 30 by 36 in. The jig tailings go to the tailings pile or railroad cars for ballast. The screen discharge goes to concentrate bins and the hutch to a Diester screen making two sizes. The coarse goes to one Diester table and the fines to another. Table tailings go to waste and the middlings are returned to the screen.

The National Pigments and Chemical Co. consumes its entire output in its own products and operates its washing plants to suit its own requirements rather than the market specifications. Some of these products are manufactured in buildings adjacent to the washing plants. The rest of the barite is shipped to the St. Louis plants.

The Superior Mineral Co. washing plant at Cadet is driven by purchased electric power. It has a capacity of about 500 tons per day. Ore is brought from the electric shovel in 5-yd. cars and dumped onto the "bull pen" or grizzly floor which is formed by railroad rails 6 in. apart. Here boulders, roots, etc. are separated by hand and thrown out. The material passing the grizzly goes to two 30-ft. double-log washers. The clay and water goes to the mud pond and the washed barite and rock to a drag delivering to a small grizzly. The oversize goes to a jaw crusher, then, with the undersize, is delivered by elevator to a steel-plate trommel with ⅝-in. round holes. The oversize from this generally goes to waste, but if necessary may be returned to the crusher. The through product goes to a storage bin. This delivers by an apron feeder to a vibrating screen of about 5 mesh. The oversize goes to three Woodbury jigs in series; the fines to a revolving screen. The first two cells make clean concentrates from the screen. The discharge from the third goes to a special scrubber, the cleaned concentrates from which go to the concentrate bin, and the fines and tailings join the fines removed from the jig feed.

The fines from the 5-mesh vibrating screen, the hutch discharge of the jigs and the fines from the scrubber join and are sent to a trommel.

The oversize from this goes to a small two-cell Woodbury jig making clean concentrates and tailings. The hutch discharge and the fines from the trommel go to a drag dewaterer and the thickened discharge to a Fahrenwald classifier. The overflow of the dewaterer goes to a settling box with a spigot discharge feeding a Plato table, from which the overflow goes to waste. The spigots of the classifier feed four Plato tables. The concentrates from these go to shipping bins; tails to waste; the middlings return to the dewaterer.

Table concentrates are generally higher grade than jig concentrates. Clean table concentrates often will analyze 98 per cent. barium sulfate, 0.25 to 0.50 per cent. ferric oxide, and 0.5 to 1.0 per cent. silica.

In the Central district individual deposits are comparatively small, so the washers are small and of a portable nature. Lack of available water is also a factor in this district, necessitating pumping a considerable distance and against a considerable head. When a washing plant can be employed it greatly increases the yield possible by hand methods.

As an example of this, a pit was worked about 30 years ago by hand methods and the present dimensions of the excavation indicate that about 30,000 tons was removed from which the barite was hand-sorted and shipped, but no records of shipments are available to show the recovery. Recently a small washing plant has been working the old dump. When half through, 2000 tons of barite had been recovered, indicating a potential recovery of 4000 tons from the dump of 30,000 tons, or about 13 per cent. left in the dirt by hand-sorting methods.

The plant of the Versailles Mining Co. near Bagnell is typical of the plants that have been working on some of the deposits and old dumps left by hand mining.

Horse-drawn drag scrapers move the dirt from the old dump to a dumping platform above the washer, which has a capacity of 150 to 200 tons of dirt per 10 hr. A jet of water flushes it into a launder leading to the log in the plant, which has a single 18-ft. log. Overflow from this goes to waste, and the washed ore goes to a 10 by 14-in. jaw crusher. A bucket elevator delivers the crushed ore to a trommel covered with a punched plate screen (5/8-in.) with an outer screen of woven wire with 3/32-in. openings. The oversize from the 5/8-in. screen goes to 12 by 20-in. rolls and back to the elevator and screen. The 5/8-in. plus 3/32-in. product feeds a four-cell Harz jig (cells 30 by 42 in.). The draw-off from the screens is clean concentrates. Some of the hutch discharge is clean and some is returned to the circuit.

The product passing the 3/32-in. screen feeds one concentrating table making concentrates and tailings. No attempt is made to separate and return a middling product. The washer is driven by a 50-hp. gasoline engine. In addition, a 25-hp. gasoline engine is used to operate the water-supply pump delivering 300 gal. per min. against a head of 150 feet.

MARKETING

Some consumers have their own mines and washing plants for the production of their own needs. Most of the production, however, is sold in the open market or on a contract basis. Numerous buyers are in the districts, especially in Washington County. They may buy for one consumer or operate on a brokerage basis, buying in small lots from the smaller producers and selling to the market that offers the best outlet at the time. These buyers maintain docks or loading platforms at various shipping points where the ore is accumulated. It is sometimes resorted and mixed to make a uniform grade. It is always loaded by forking, to make a final elimination of adhering dirt and clay.

The larger producers, unless consuming their own output, sell direct to the consumers without the intermediary buyer. The buyer has his legitimate place, however, as many small producers of hand-mined ore turn out only a few tons a year during the slack farming season and there must be some agency to accumulate these small lots into carload shipments.

Sampling of shipments is rarely done at point of origin. It is usual to take the consumer's sampling and analysis. Deductions are made for moisture, especially on washed ore.

Most of the Missouri ore is marketed in St. Louis, Illinois and the Ohio Valley.

There are no fixed standards of quality or grade, although there is a generally accepted grade for the lithopone trade and one for other uses, but penalties and bonuses on this standard often vary with the individual contract or sale.

The usual standard for lithopone is a minimum of 93 per cent. $BaSO_4$ with not over 2.5 per cent. combined Al_2O_3 and Fe_2O_3 and a penalty of 10 c. per ton for each 0.1 per cent. Fe_2O_3 over 1 per cent. Sometimes a bonus is paid for a $BaSO_4$ content greater than 93 per cent. One contract paid a bonus of 10 c. for each unit of $BaSO_4$ over 93 per cent. and also a bonus for iron oxide of less than 1 per cent.

Shipments seldom fail to meet specifications. Table 2 gives an analysis of several carloads of hand-picked ore and washed ore from the Central district.

TABLE 2.—*Analysis of Ore from Central District*

	Minimum		Maximum		Average	
	$BaSO_4$, Per Cent.	R_2O_3, Per Cent.	$BaSO_4$, Per Cent.	R_2O_3, Per Cent.	$BaSO_4$, Per Cent.	R_2O_3 Per Cent.
Six cars hand-mined barite	96.28	0.90	97.47	2.10	96.43	1.59
Eight cars washed barite	93.00	0.62	95.22	1.30	94.21	0.85

TRANSPORTATION

The construction of good highways has greatly extended the area available for mining. In some cases ore is hauled as much as 20 miles to the railroad. The cost of one 20-mile haul is $3 per ton.

One 8-mile haul was $1 per ton. Both of these were by motor truck. Loading from the platform or dock averages 15 c. per ton.

Both districts are afforded good rail transportation by the Missouri Pacific R. R., giving a direct outlet to St. Louis and through the St. Louis gateway to the north and east.

Principal railroad shipping points in the Washington County district are Potosi, Mineral Point, Cadet, Blackwell, Tiff and De Soto; in the Central district, Bagnell, Eldon, Russellville, and Versailles.

UTILIZATION OF BARITE

Barite has a great diversity of uses. These may be classified into two general groups: (1) those in which it is used as natural barite after proper mechanical treatment; and (2) those in which it is the source of barium for various chemical compounds in turn used for a variety of purposes.

Use in Natural State

When used in its natural state it always is prepared by grinding to a fine state of division and also sometimes by bleaching with sulfuric acid. In this form it is used as a filler in some rubber articles, in paper, paint, oilcloth, molded goods, such as buttons, in suspension in liquids in oil and gas-well drilling, as a flux in brass melting, as a component in a special glass, etc.

As a rubber filler, bleached or unbleached, it is employed where a soft rubber article is desired, as in rubber bands, tubing and the like, but not where strength and resistance to abrasion is required, as in auto tires. A considerable amount of unbleached barite is used in molded goods such as buttons, hard rubber articles, and probably phonograph records.

The bleached form is used in paper where weight is desirable with a high finish, as in playing cards and plate paper. Natural barite is not often used in high-grade paints. A small amount of the bleached is used direct as an extender and loader and as a base for lake colors.

Extremely finely ground barite mixed with a suitable suspending medium has been found effective in making up heavy liquids used in drilling for oil and gas. The heavy liquid tends to hold back the gas flow until casing can be placed. Finely divided iron oxide in suspension has also been employed for the same purpose. The use of barite with a suspending medium is patented by the National Pigments and Chemical Co., and the compound is made at their plant near Cadet. It is marketed under the name of "Baroid."

A new use is as a component of certain glass mixes. A very low iron barite is required, of course. It is claimed to be especially good in glass for blowing or molding, allowing sharp corners and intricate patterns to be completely filled. Its use for this purpose is patented.

Various grades of ground barites, bleached and unbleached, are prepared at two plants in St. Louis, The National Pigments and Chemical Co. and C. P. DeLore.

As a Source of Barium

Barite is the source of barium for the manufacture of lithopone, *blanc fixe,* titanium paint pigment and various barium salts and compounds such as the carbonate, chloride, sulfide, nitrate, the oxides and hydroxide.

According to figures of the United States Bureau of Mines for 1927, of the crude barite, domestic and foreign, consumed in the United States, 65 per cent. was used in lithopone, 24 per cent. in ground barite, and 11 per cent. in barium chemicals.

Lithopone is a white pigment made by the coprecipitation of zinc sulfide and barium sulfate from solutions of zinc sulfate and barium sulfide. It contains approximately 70 per cent. barium sulfate and 30 per cent. zinc sulfide. Usually 1 to 5 per cent. of the sulfide is replaced by zinc oxide. Its principal use is in paints, especially for inside finish. It is also largely used as a filler in rubber, where its extremely fine grain is of advantage.

Blanc fixe is precipitated barium sulfate and differs only from natural barite in its purity, extreme whiteness and fine grain size, made possible by proper control of the precipitation. It is used in paints and as a paper and rubber filler in the same way as native barite, but where a superior product is required.

Titanium pigment or titanium white[5] is a mixture of 75 to 65 per cent. of barium sulfate and 25 to 35 per cent. of titanium oxide. Many patents have been taken out covering its method of preparation. A superior product of the proper opacity and physical properties can be made only by careful control of conditions of manufacture. It makes a paint pigment of extreme whiteness and opacity, does not discolor with age, and is nonpoisonous. It is manufactured in St. Louis by the Titanium Pigment Co. from Washington County barite and is marketed under the trade name of "Titanox."

Barium Salts.—The carbonate is perhaps the most widely used of the barium salts. Its two principal uses are in the ceramic industry, where it is mixed with certain clays to prevent "scumming" or efflorescence on the fired ware, such as face brick, and in water purification, especially waters containing soluble sulfates that are to be used for boiler purposes.[6] It is also used as a rat poison. In the preparation of other barium salts the sulfide prepared by the reduction of the sulfate is the usual starting point. Barium chloride is used in the leather industry and to some extent in boiler compounds. Barium sulfide has a small direct use in vulcanizing and weighting rubber. The nitrate is used in pyrotechnics. The dioxide is used as a bleaching reagent and in the manufacture of hydrogen peroxide. The hydroxide is used in sugar and petroleum refining.

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[5] Titanium White. *The Mining Jnl.* (1925) **149,** 271. Titanium White Pigment. *Chem. Age* (1926) **34.**

Acknowledgments

The author wishes to thank the many operators in the district for the cordial assistance given in the collection of information and especially acknowledges his indebtedness to A. E. Stocking, W. C. Wolf, and H. B. Hart.

DISCUSSION

MEMBER.—How far is it from these deposits to the nearest bodies of igneous rocks?

W. F. POND, Nashville, Tenn.—As shown on the State geological map of Missouri, there are knobs of porphyry about 12 miles west of Potosi and about the same distance southeast of Potosi, which is the approximate center of the southeast Missouri barite area. This porphyry is the same as that which is exposed extensively to the southeast and has been demonstrated to be of Pre-Cambrian age. Deep drilling in the area has shown the normal succession of sedimentary rocks. There is no direct evidence of intrusive igneous activity in this district.

W. T. SCHALLER, Washington, D. C.—Could the original source of the barium be referred to a barium feldspar, either celsian or hyalophane, distributed through some rock? A rock specimen from Alaska consisted of pyroxene, celsian and gillespite (barium, iron, silicate), which in its mineral composition and structure seemed to be the result of contact metamorphism, suggesting the possible existence of a rock analogous to limestone but consisting of barium carbonate.

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[6] C. J. Rodman: Water Softening with Barium Salts. *Chem. & Met. Eng.* (1928) **35**, 221.

Clay Mining for Quality

By H. E. Nold,* Columbus, Ohio

(New York Meeting, February, 1929)

This paper is an effort to explain in a simple manner the fundamental principles involved in examining a clay deposit for both quantity and quality and in operating a clay mine, either open-pit or underground, so that the best quality of finished material can be made from the raw material available. The writer is aware that what is recommended in this paper is everyday practice in many refractories plants and in some other heavy-clay products plants, but observations made while visiting practically every clay mine in Ohio have convinced him that the practices herein discussed are not as widely used as their importance warrants. It is true that the problem of mining for quality is more acute in refractories plants than in most heavy-clay products plants, yet some of the latter are finding it profitable to pay more than ordinary attention to the selection and blending of their raw materials. It is hoped that this short paper may prove helpful to some who are now finding it difficult to produce a uniform product of as high quality as their clay deposit seems to warrant.

The coordination of the work of the clay mine or pit with the manufacturing plant, to the end that the best product possible be made from the raw material available, presupposes a rather intimate knowledge of the clay deposit. The knowledge must include not only information of the amount of clay available but also rather detailed information about both vertical and horizontal variations in quality of the clay. This information should cover three things: quality, quantity, and location.

SAMPLING UNDEVELOPED DEPOSITS

In the case of an undeveloped deposit with a shallow overburden, pitting may be the cheapest way of testing. For thick deposits or deposits with rather heavy overburden, the diamond drill, using a double core barrel, offers the best means of testing. When using the diamond drill in clay testing, it is doubtful whether it pays to take less than 2-in. cores; it is difficult to get complete cores with smaller drills, and the quantity of material obtained with smaller cores is not sufficient for the necessary tests.

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The number of drill holes necessary to prove the quantity and quality of a clay deposit varies with the characteristics of the deposit. The more regular the deposit, the fewer the holes necessary; and conversely, the greater the variations within the deposit, the more holes necessary. The greatest amount of information for the fewest feet of drilling, and therefore at least cost, can usually be obtained by systematic location of the drill holes. It is good practice to locate them on the corners of squares. The system of location of the holes having been decided, money can frequently be saved by drilling a few rather widely separated holes first rather than by drilling them in order as they appear on the projected plan. For example, one hole in each quarter of the property may demonstrate the worthlessness of the entire property as conclusively as holes more closely spaced, or the drilling of only part of the holes may prove the deposit to be so regular and uniform that more closely spaced holes are unnecessary. By this system, the spacing of the drill holes may be made smaller and smaller by locating the new holes between the holes already drilled. In one case, for example, a hole every 1200 ft. may give all the information necessary while a hole every 300 ft. or even oftener may be necessary if the deposit is quite irregular.

Sampling the Mine

While mining the deposit, systematic sampling of all the working faces may be necessary. The face samples, after testing, should give such detailed information that they can be used as a basis for both blending and selecting clays. The quality of clay in horizontal bedded deposits may vary greatly and very abruptly in a vertical direction but usually varies more gradually in a horizontal direction. Face samples should represent the clay as mined. Their horizontal distribution throughout the mine or pit should be decided by the variation of the clay from place to place but should always be taken so frequently that collectively they truly represent the clay as mined. In case of doubt, it is better to cut too many samples rather than too few. It is common practice in some mines producing refractory clays to sample each working face once a month, which means a sample every 60 to 100 ft. in each direction throughout the mine.

Tests made from samples, no matter how carefully made, are no more accurate or reliable than the samples from which they were made. A few lumps selected from the clay shot down at the working face are practically worthless as samples. A few lumps knocked off the face at various places are also of little value as samples. A sample, to be valuable, should represent the clay from the entire face where taken, each part in the same proportion that it occurs in the unmined face. For control of mining each sample must be labeled so that its location in the mine can be told exactly.

The larger the sample taken, at any working place, the easier it can be made to be truly representative. Large samples, however, are expensive to take and test and are therefore undesirable. The best way to get a truly representative small sample is to cut a uniform channel from top to bottom of the deposit. The place to be sampled should be carefully cleaned; all loose material should be removed and fresh clay should be exposed. The size channel recommended for both usefulness and convenience is 4 or 5 in. wide and 1 or 2 in. deep. It is important that the size of the channel be kept uniform from top to bottom of the deposit. It is important, but sometimes difficult, to keep the size of the channel the same in the hard parts of the deposit as in the soft parts. The sample, as cut, may be too large to take to the testing laboratory; if so, it should be crushed and quartered to the desired size at once, in accordance with the standard practice. This practice is described in many articles and papers on sampling and need not be discussed here. The prepared samples should be placed in hermetically sealed cans for transference to the testing laboratory if moisture determinations are to be made; if moisture is not needed they can be handled more conveniently in small canvas bags. When cutting the sample, it is important to prevent material from other parts of the face from falling into it. The writer has found it convenient to cut the sample either onto a large canvas spread on the bottom in front of the face being sampled or into a canvas bag kept open by a wire and held against the face immediately below the place from which the sample is being cut.

The channel across the deposit should be divided into as many samples, vertically, as may be necessary to show the properties of the several varieties of clay. The number of samples necessary can usually be told by the appearance and condition of the clay; if this is not a safe guide, it is necessary to divide the channel arbitrarily into as small portions as may be necessary to localize the various impurities and grades of clay. For example, the upper 3 ft. of the deposit might be included in one sample and each succeeding 2 or 3 ft. be made a separate sample. This breaking of the channel into several samples is necessary so to locate the varieties of clay and the impurities that the miners can discard the undesirable parts with certainty in mining. The practice in some refractory plants in Ohio is to determine the least thickness of clay that can be economically mined and, when cutting a channel sample, this thickness is taken as the length of the first sample cut from the top downward. Below this as many samples are taken as the appearance of the clay indicates are necessary. If tests of the top sample are not satisfactory, that part of the mine or pit is abandoned.

A continued careful comparison between the results of the tests and the appearance of the clays tested frequently makes it possible to recognize all the varieties of clays in any deposit with such a degree of certainty that regular channel sampling can be dispensed with. In some refractory plants it is the regular practice to examine each car of clay before and during dumping. If the car contains undue amounts of recognized undesirable materials, it is set aside and the miner who loaded it is called from the mine and required to pick out the impurities on his own time. Lesser amounts of impurities are discarded by the dumper. The picking table, long used in coal mines, has lately been installed in some clay plants to aid in removing impurities. If the car contains material of unknown or doubtful value, this material is sampled, and, if necessary, the face from which the car came is sampled. The value of the doubtful material is determined and mining instructions are given according to the results of the tests. This method should prove satisfactory if the clay is well known and there is a distinct difference in appearance between the good and poor material.

Clay Testing

Proper and sufficient samples having been obtained by boring channeling or by both methods, it then becomes necessary to develop tests which can be made rapidly and which can be coordinated with plant practice. For this a testing laboratory equipped with grinding and screening equipment and a small test furnace or kiln is necessary.

In order that the results of such tests may be used effectively as a guide and aid in mining, the tests should be completed and the results reported to the mine superintendent or foreman within 48 hr. after the time the samples are taken at the face.

The writer is not a ceramic technologist and is not competent to discuss or suggest the details of testing the samples for any given class of ware, but he does know that satisfactory testing of mine samples, within the time limit mentioned, is being done in some refractory plants as a matter of regular routine.

Selective Mining

The sampling and testing having been satisfactorily done, the management is now in position to give instructions to the mine superintendent which will enable him to properly select the clay in each part of the mine sampled. The writer knows of one company manufacturing refractories where the results of regular mine sampling and testing are transmitted to the mine foremen in the form of written instructions telling exactly what to mine and what to discard at each working face. Here any change in the appearance of the clay in any part of the mine is immediately reported to the testing department and a man is detailed to resample such place or places. New mining orders are issued on the basis of the new tests.

In order that the selective mining based on sampling and testing may be effectively carried out, it is necessary to do more than give orders to the mine officials. These men can cooperate more wholeheartedly and intelligently if they know what it is all about. They should know more about the clay processing and firing than many of them now know. Take them into the manufacturing plant and show them the effects of poor clay on the finished product. If the mine officials understand the importance of the tests and of carefully selecting the clay in the mine or pit according to tests, the battle for well-graded clay is at least half won (assuming loyal and competent mine officials). They can and will then sell the idea of careful mining to the men loading the clay. Most men are willing to cooperate if they know what it is all about and without the wholehearted and intelligent cooperation of both mine officials and miners selective mining of clay will be a failure.

After control sampling and testing is once established and the mine officials and the miners are educated as to the plant needs, selective mining for quality in either open-pit or underground mine can be carried on to practically any degree desired, provided the management realizes that the more careful the selection made by the miners the less the output of the miner per day, and consequently the higher the cost per ton of the clay. The position and distribution, in the deposit, of the material to be discarded in the mine has much to do with the cost of selective mining. There are many places where the different varieties of clay are so distributed by nature that they can be shot and loaded separately with little inconvenience to the miner, while in other deposits they are so intermixed in the deposit that a slow hand and shovel sorting are necessary if any appreciable degree of selection is to be accomplished. It should be recognized that mechanical loaders (power shovels and other types) do not lend themselves to a very great degree of selective mining and that such machines must be replaced by hand loading if a great degree of selection is desired.

Assuming a sufficient knowledge of the clay, the degree of selective mining to be done should be regulated by what is profitable. As long as the cost of selective mining is more than counterbalanced by increased value of the finished product or by decreased losses in manufacture, selective mining is profitable, but when the cost of further selection of clays is no longer reflected by a corresponding increase in value of the finished product, the economic limit has been reached. This is a simple economic problem to be solved by the management of each plant and mine. In solving this problem, however, one must not lose sight of the fact that the entire cost of selection in the mine, particularly where such selection means the abandoning of certain parts of the mine, may not immediately be reflected on the cost sheets because the cost of leaving the clay in certain parts of the mine may show up at a later date in the form of decreased recovery, increased drainage and haulage costs, etc. In this connection, it is important that the management know something of the problems of mine operation in order that they may demand from the mine only that which is fair and reasonable. The attitude in too many Ohio plants, particularly those using clay from underground mines, is that the mine is a necessary evil and the problems of mining are not understood and little studied by the management. There is all too little appreciation of the fact that economies that would make possible considerable increases in the annual dividends are possible in many of our clay mines and pits. When the manufacturer of refractories and heavy-clay products is as anxious to discuss his mine problems and economies in mining as he is to discuss his forming, drying and firing problems, real progress will be made in mining. A dollar saved in the mine or pit is worth just as much as a dollar saved in the plant, and a dollar wisely spent in the mine is just as good an investment as a dollar wisely spent in the plant.

Blending and Mixing

Another element of great importance is the proper blending of the various varieties of clay coming from the mine, to the end that the plant may make a product of uniform quality. Some of the plants have large storage capacities into which the clay from the primary crusher is distributed in horizontal layers either by means of belt conveyors with mechanical trippers or otherwise, and from which the clay is fed to the dry pans or wet pans in as nearly vertical slices as practicable. With a storage capacity of not less than two days run of the plant, this method is producing an apparently well blended and quite uniform product in a number of plants.

In many plants, particularly the smaller ones, the clay from the mine or pit is dumped directly into a small hopper and thence directly to the dry pan and onto the forming equipment, with very little storage capacity or opportunity for mixing or blending anywhere in the process. The writer's observations in both open pits and underground mines have convinced him that blending and mixing in the mine so as to produce a uniform product with the flow sheet described is impossible unless the clay is exceptionally uniform in quality throughout the deposit. There is no doubt that frequently manufacturing troubles attributed to other causes are in fact due to lack of proper blending and mixing of the clays before they are fed to the forming machines.

The writer has visited a few open pits, using power shovels, where blending, satisfactory for heavy-clay products manufacture, was apparently being obtained. This was done by the shovel operator, who mixed and prepared the clay while the cars were being dumped. Such mixing is beyond the scope of hand loading. In another pit recently visited, the deposit is in the form of a main lower bench of one quality and an upper and thinner bench of another quality. In this pit the lower bench was loaded by steam shovel while the upper bench was loaded by hand, a measured quantity being dumped into each pit car. A number of plants depend on the shooting to mix the clay so that it can be loaded by hand into mine cars and dumped directly to the dry pan. Personal observations have convinced the writer that shooting does not mix the clay to the extent frequently believed and that satisfactory blending by hand loading is not practical unless the clay is so uniform as not to need much blending. The clay is usually shot in benches and the final pile is in layers essentially corresponding to the benches shot. The average mine or pit can produce clay so that the average of one day's run will agree with any other day's run, but few mines or pits can produce clay so that each mine car throughout the day will have the same quality of clays. The mixing and blending of clays is essentially a plant job and not a mine job.

Summary

Summarizing, we may say:

1. An intimate knowledge of the varieties of clay and impurities in any clay deposit based on sampling and testing is necessary before selective mining can be satisfactorily done.
2. Tests of samples must be coordinated with plant practice.
3. Results of tests of samples should be reported to the mine officials within 48 hr. after taking the samples.
4. The mine officials should know more than they do about the effects of poor clay and impurities on the finished product.
5. The management should be more familiar with mining conditions and problems.
6. Selective mining is physically possible but increases costs according to the degree of selection required and the condition of the clay in the deposit.
7. The degree of selection to be required is an economic problem to be solved by the management of each plant.
8. The blending of clays for uniformity of product is essentially a plant problem and not a mine problem.

DISCUSSION

A. F. GREAVES-WALKER, Raleigh, N. C. (written discussion).—Professor Nold's paper touches on an important phase of the manufacture of clay products that is too often overlooked or slighted. In discussing the subject of sampling undeveloped deposits, Professor Nold does not touch on the sampling of those deposits in which the strata have been folded and therefore dip at angles ranging from vertical almost to horizontal. In some parts of the country such deposits are common and a great deal of money has been wasted in drilling. Vertical drill holes in such cases are extremely misleading to those without geological training. Such deposits should be examined by trenching on the surface at right angles to the strike, care being taken to trench deep enough to get below the weathered surface. Ordinarily little is gained by further examination but additional information can be gained from faces exposed in gullies or stream valleys, which usually occur in the vicinity.

Some Properties of Fuller's Earth and Acid-treated Earths as Oil-refining Adsorbents*

By C. W. Davis† and L. R. Messer,‡ Reno, Nev.

(Los Angeles Meeting, September, 1928)

The name fuller's earth, which was derived from its early use in "fulling" or removing grease from woolen goods, is a term that is generally considered to designate mineral matter, containing hydrous aluminum silicates of the clay group, which has a high capacity for removing colors from oils, whether of animal, vegetable, or mineral origin. Fuller's earths are usually dried and may be "tempered" by heating or may be treated with water or reagents to improve their efficiency.

For our purpose the term "acid-treated earths" will include those products from the partial decomposition of mineral water by acid which are equal to or more efficient than an untreated fuller's earth from Quincy, Fla., when used in bleaching mineral oil by the "contact" process.

In general, the contact process for treating oils consists of heating and agitating a mixture of oil and finely divided adsorbent, the latter being subsequently removed by filtration.

The term "adsorbent" is used because, although there has been a great deal of controversy over the mechanism of removal of impurities from oils by mineral matter, there is no doubt that adsorption, which involves the surface of the solid, plays a major part.[1]

The properties that make fuller's earth useful in the treatment of petroleum products by "percolation" and in the purification of edible oils by the contact process have long been studied, but the comparatively recent adaptation of the contact process to the bleaching of petroleum products has made desirable the further investigation of fuller's earth and other adsorbent earths.

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* This paper represents work done under a cooperative agreement between the U. S. Bureau of Mines and the University of Nevada. Published by permission of the Director, U. S. Bureau of Mines.

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[1] H. L. Kauffman: The Role of Adsorption in Petroleum Refining. *Chem. & Met. Eng.* (1924) **30**, 153.

B. Neumann, and S. Kober: Bleaching Action of Bleaching Earths on Oils. *Ztsch. angew. Chem.* (1927) **40**, 337; *Chem. Abs.* (1927) **21**, 2811.

T. H. Rogers, F. V. Grimm and N. E. Lemmon: Adsorption Studies on the Decolorization of Mineral Oils. *Ind. & Eng. Chem.* (1926) **18**, 164.

This paper presents some of the results of work performed in a study of adsorbents used in the contact process for bleaching oils.

About 40 earths, including pure clay minerals supplied by the U. S. National Museum, various types of fuller's earths, acid-treated earths, and impure clays, were used in the tests. The following 17 earths, of which the first 14 require acid treatment to develop their bleaching power, were selected for special attention:

No. 1. Calcareous earth from Moapa Valley, Nev.
No. 2. Commercial bleaching earth from Southern California.
No. 3. Bleaching earth from Beatty, Nev.
No. 4. Commercial bleaching earth from near Shoshone, Calif.
No. 5. Commercial bleaching earth from Southern California.
No. 6. Commercial bleaching earth from Southern California.
No. 7. Otayite from Otay Valley, Calif.
No. 8. Shoshonite from Shoshone, Calif.
No. 9. Calcareous earth from Las Vegas, Nev.
No. 10. Bentonite from Lahontan Dam, Nev.
No. 11. Bentonite from Clay Spur, Wyo.
No. 12. Bentonite, commercial adsorbent, from Southern California.
No. 13. Bleaching earth from Acme, Nev.
No. 14. Montmorillonite, pure mineral, from San Diego, Calif.
No. 15. Halloysite, pure mineral, from Horse Cave, Ky.
No. 16. Commercial fuller's earth from Quincy, Fla.
No. 17. Commercial fuller's earth, high-magnesia type, from Nye County, Nev.

The color of oils was measured by the Ives tint photometer, which gives a quantitative measure of the amount of each color for which it is desired to test. Determinations of the red, yellow, and green content were found satisfactory for our work, as red and yellow are sufficient for most tests (color glasses of the red and yellow series with the Lovibond tintometer have been used in the past when an accurate measure of the color of oils was required). The instrument is graduated from 0 to 100 in such a way that the lighter the color of the oil becomes the higher is the reading. A reading of 100 indicates a colorless oil.

ORIGIN OF FULLER'S EARTHS

Fuller's earth deposits are described as Tertiary sedimentary beds, with the exception of an occurrence in Arkansas[2] that results from alterations *in situ* of basic dikes containing a high percentage of ferromagnesium minerals. Deposits from South Dakota[3] and Illinois[4] are found interbedded with clays and sandy beds. Bedding in the fuller's earth is not well developed and the outcrops exhibit a concretionary or nodular structure which when broken down into a talus forms conchoidal or shell-shaped fragments.

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[2] H. D. Miser: Developed Deposits of Fuller's Earth in Arkansas: U. S. Geol. Survey *Bull.* 530 (1912) 10.

The Georgia and Florida fuller's earth is thought to have been deposited as a calcareous clay in shallow marine waters. Shearer says: "The leaching of the calcium carbonate left a large volume of openings while the silica originally present, together with that deposited from solution, left a framework strong enough to hold the pores open."[5]

Quaternary deposits are found in New York[6] and in Massachusetts.[7] The deposit in Massachusetts is glacial silt, free from pebbles and containing only a few large boulders.

California bleaching earth of the high-magnesia type seems to have been produced by the alteration of a crystal tuff[8] that contained some rock fragments. The filling between the fragmental material was a fine volcanic dust. The tuff is believed to have been basic because of the notable magnesium content shown by chemical analysis, the absence of quartz, the basic nature of the feldspars, and the iron present. Thin sections show rock fragments, numerous crystals of fairly basic plagioclase, and occasional small particles of hornblende. The interstitial filling is an aggregate of fine microscopic fragments and altered material which is light brown and turbid when seen by transmitted light. Under crossed nicols this groundmass shows faint polarization in grains too small to be identified individually by their optical properties. In the groundmass are seen veinlets and irregular patches of a transparent, isotropic, light brown substance which has an index slightly higher than balsam and is thought to be gelatinous silica formed during the alteration of the tuff. The rock fragments are light brown and contain some broken crystals of feldspar. Under crossed nicols the isotropic, light brown material is seen to contain a few small spherulites. In most instances the borders of the fragments show polarization due to crystallization. The feldspar crystals are essentially unaltered, appearing practically fresh. Most of the hornblende has been altered along with the groundmass. Other common minerals are specks of iron oxide and opaline silica. The samples from which slides were made were soaked for 48 hr. in a partial vacuum in bakelite, which entered into the material along occasional small openings and seams, but in such a manner as to indicate that there were not enough openings to allow the bakelite to saturate the sample.

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[3] H. Ries: Fuller's Earth of South Dakota. *Trans.* A. I. M. E. (1897) **27**, 333.

[4] J. E. Lomar: Preliminary Report on the Fuller's Earth Deposits of Pulaski County, Ill.: State Geol. Survey *Rept. of Investigation* 15 (1928) 10.

[5] H. K. Shearer: Bauxite and Fuller's Earth of the Coastal Plain of Georgia. Geol. Survey of Ga., *Bull.* 31 (1917).

[6] H. Ries: Fuller's Earth: New York State Mus. *Bull.* 35 (1900) 8.

[7] W. C. Alden: Fuller's Earth and Brick Clays near Clinton, Mass. U. S. Geol. Survey *Bull.* 430 (1910) 402.

[8] L. V. Pirsson defines crystal tuff as compacted, fragmental, volcanic material in which crystals of individual minerals are conspicuous. [The Microscopical Characters of Volcanic Tuffs. *Amer. Jnl. Sci.* (1915) **50**, 196.]

If the California bleaching earth was produced by the alteration of a basic tuff, alteration products from other basic tuffs should have the property of bleaching oils. With this idea in mind a sample of tuff was obtained from an exposure along the Little Truckee River 1½ miles north of Boca, Calif. This tuff when dried and ground to pass 200 mesh was found to bleach mineral oil fairly well though not well enough for commercial purposes. A hand specimen of this tuff was buff-colored and considerably weathered, but it decidedly resembled tuff rather than clay. Thin sections show it to be a somewhat altered basaltic lithic tuff containing a few fragmental crystals, much iron oxide, and some altered crystals of olivene. The groundmass is also brown and under crossed nicols looks very much like the groundmass of California bleaching earth.

Slides were prepared of California earths which, when acid-treated, produce commercial absorbents. These earths can be classed as those which show the texture of volcanic ash and those which are so fine grained that the texture before alteration can not be determined.

Thin sections of earths which have had a pyroclastic origin show outlines of lithic or glassy fragments. One section of otaylite is composed wholly of glass shards, under 0.3 mm. in length, and is a typical vitreous tuff similar to that illustrated by Pirsson.[9] However, in this otaylite section the glass shards are almost completely devitrified. The larger fragments have fine, fibrous, radial crystals around the borders. Small globules of glass, not over 0.15 mm. dia., have a similar arrangement, with crystals radiating from the center, which under crossed nicols appear exactly like spherulites. These circular structures were probably formed upon the devitrification of the glass, and it is proposed that they be called pseudospherulites.

In the slides of fine-grained earths it was impossible to discover the outlines of glass particles by removing the upper nicol, lowering the condenser, shading the mirror, and moving the objective slightly. However, pseudospherulites were found in one slide of montmorillonite. Under crossed nicols with high-power objective the field appeared to be filled with minute laths of a micaceous mineral having nearly parallel extinction, negative elongation, and moderate birefringence. Fresh powders of montmorillonite gave a mean index of 1.505 ± 0.003. The slides of two other fine-grained commercial earths appeared like montmorillonite in patches, although other light brown areas which were isotropic showed incipient crystallization. The fresh powders of these earths gave a mean index of 1.495.

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[9] L. V. Pirsson: *Op. cit.*

Earths which commonly require acid treatment before becoming effective as adsorbents were probably formed by the alteration of volcanic ash deposited as vitric tuffs which were later devitrified. It is suggested that during alteration the deposits were subjected to attack by alkaline solutions formed from soda, potash, or lime in the tuff. Under these conditions hydrous aluminum silicates were formed which show a high silica-alumina ratio.

Tests were made with clay minerals that have been thought to be the cause of the bleaching action of acid-treated earths. Pure clay minerals were obtained from the U. S. National Museum. Montmorillonite, halloysite, pyrophyllite, and kaolinite were given different acid treatments and the products were contacted with oils. Of these, only montmorillonite gave a product that compared at all favorably with commercial products.

General Character of Adsorbents

Fuller's earth is nearly always described as a hydrous aluminum silicate, the chemical composition of which may be too variable to afford a means of identification; as shown in Table 1, the analyses of various samples of fuller's earths (A and B) differ markedly, and various clays (C) having no oil-bleaching value may approximate the composition of good bleachers.[10]

In a general way, however, the compositions of fuller's earths are informative, for their ratio of silica to alumina is much higher than that of kaolinite (being about four, five, or six to one for good grades of fuller's earth) and in the highly efficient bleaching agent occurring in Nye County, Nev., the composition is that of a hydrous magnesium silicate[11] (analysis D, Table 1).

The earths from which good acid-treated oil-bleaching adsorbents may be prepared are, like fuller's earths, variable in composition, but their silica-alumina ratio (generally two or three to one), although greater than that of kaolin, is not so high as that of fuller's earth (analyses E and F, Table 1). After acid treatment the silica-alumina ratio may be five or six or more to one, approaching or even exceeding that of fuller's earth (analysis G, Table 1).

When fuller's earths, containing no soluble salts, have a high apparent acidity in water, they invariably have good bleaching action; most fuller's earths from England and from the Coastal Plain of the southeastern

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[10] C. L. Parsons: Fuller's Earth: U. S. Bureau of Mines *Bull.* 71 (1913); see also H. K. Shearer, *Op. cit.*

[11] V. M. Arciniega: Refinery Clays Mined in Nevada. *Eng. & Min. Jnl.* (1926) **121**, 408.

United States have this property to the extent of requiring from 10 to 150 c.c. of N/10 sodium hydroxide solution to neutralize the apparent acidity from 100 g. of earth, using phenolphthalein as an indicator.[12]

TABLE 1.—*Analyses of Adsorbents and Clays*

	A	B	C	D	E	F	G
SiO_2, per cent	72.95	58.10	58.72	57.95	58.04	47.38	59.30
Al_2O_3, per cent	12.65	15.43	16.01	1.57	15.81	15.38	9.53
Fe_2O_3, per cent	3.56	4.95	2.12	0.85	3.10	2.57	1.70
FeO, per cent	0.47	0.30					
MgO, per cent	0.57	2.44	3.30	19.71	4.12	4.24	3.20
CaO, per cent	1.00	1.75	1.05	4.17	1.19	2.25	1.13
Na_2O, per cent	0.20	0.27	2.11	1.84	1.62	0.59	0.40
K_2O, per cent	0.68	0.66	1.50	0.43	0.75		
CO_2, per cent		0.84		1.22		None	None
(Loss −105° C.) per cent	5.77	4.59	6.21	4.82	4.92	20.50	15.10
(Loss +105° C.) per cent	1.25	9.45	8.61	8.11	10.03	7.10	8.79

A. Commercial fuller's earth from Florida. Analyst, H. K. Shearer (*Op. cit.*).

B. Commercial fuller's earth from Florida. Analyst, H. K. Shearer (*Op. cit.*).

C. Clay from Nevada with no bleaching action. Analysis by authors.

D. Commercial high-magnesia earth from Nevada. Analysis by authors.

E. Earth from which commercial acid-treated adsorbents are prepared. Analysis by authors.

F. Earth from which commercial acid-treated adsorbents are prepared. Analyst, V. Salmi (see Footnote 14).

G. Commercial adsorbent prepared from earth "F." Analyst, V. Salmi (*Op. cit.*).

The converse is not true, however, for apparent acidity is not a characteristic of the highly efficient bleaching earths occurring in California and Nevada. All of these that we have tested have a high apparent alkalinity when placed in water, requiring from 5 to 100 c.c. of N/10 acid per 100 g. of earth to effect neutrality.

Most of the earths from which efficient acid-treated adsorbents were prepared gave a basic reaction with phenolphthalein when placed in water, so that from 10 to 150 c.c. of N/10 acid per 100 g. of earth was necessary for neutralization.

The apparent specific volume of air-dried fuller's earth from the Southwestern States has been frequently mentioned as a rough measure of its bleaching efficiency, the statement being made that if an air-dried lump of fuller's earth will float in water it is almost certain to have good bleaching power.[13] Fuller's earths from other parts of the United States and from England have a higher apparent density, but a sample of high-magnesia bleaching earth of good quality floated for a short time on water.

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[12] H. K. Shearer: *Op. cit.*

[13] H. K. Shearer: *Op. cit.*
C. L. Parsons: *Op. cit.*

The property of swelling in water seems to be peculiar to most earths that become good adsorbents after acid treatment, for all earths of this type that were tested showed some swelling. Swelling is not a sure test, however, for although acid treatment produced a good bleaching agent from a bentonite from Clay Spur, Wyo., which swells to several times its own volume when placed in water, no acid treatment was found that made a satisfactory adsorbent from a South Dakota bentonite which exhibited an equal swelling power. Samples of fuller's earth and most earths from which acid treatment failed to produce good bleaching agents showed no tendency to swell when placed in water.

Two types of impurities may reduce the bleaching action of earths——inert matter such as quartz, which merely dilutes, and impurities such as iron sulfides, calcium carbonate, or soluble salts, which may reduce the original bleaching effect or may cause a decrease in bleaching efficiency when the adsorbent is heated to "temper" or "revivify" it.

Oil-bleaching Properties

In testing adsorbents to determine their value for removing impurities from oils by the contact process, several factors must be considered.

Temperature of Contacting

The temperature at which the contact process is conducted varies with each oil and should be as low as possible to obtain a satisfactory product, for although comparatively high temperatures seem to be favorable for the adsorption of impurities from oils (perhaps due to increase of adsorption capacity of the impurities through the removal of adsorbed gas or vapors, or decreased viscosity, or to increased catalytic action at elevated temperature), contact treatment at high temperatures increases the free fatty-acid content of vegetable and animal oils and tends to darken the color of lubricating oils produced from most crude petroleums.

Four oils were used in our tests. Unrefined soya bean oil was selected because it was said to be a very satisfactory standard to use in determining the bleaching action of acid-treated adsorbents.[14] A temperature of 91° C. (the temperature of the oil contained in a beaker immersed in boiling water) was found convenient to use with this oil. Medium-grade motor oils ready for treatment with adsorbents were obtained from California, Texas, and Oklahoma. The poise viscosities of the California and Texas oils at 150° F. were 0.28 and 0.62, respectively, and the Saybolt viscosity of the Oklahoma oil was 200 at 100° F. The color of the finished Oklahoma oil is 3 N.P.A.

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[14] V. Salmi: The Chemically Prepared Absorptive Clays and Their Application in the Purification of Oils, Fats, and Waxes. Los Angeles, 1926.

The optimum temperature for contacting these petroleum oils was determined by heating samples at different temperatures while mixing in the adsorbent (a commercial acid-treated earth) in proportions of 4 g. of adsorbent to 100 c.c. of oil. The heating and agitation were continued until frothing ceased (about 15 min.), as recommended by various investigators,[15] and then the oil was filtered at once through filter papers contained in heated funnels. The ratio of adsorbent to oil was chosen so as to give an average bleach, the resulting oils being lighter than the market product for the Texas oil but not quite so light as the market product for the other motor oils.

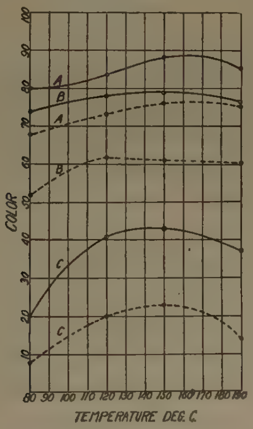

FIG. 1.—EFFECT OF CONTACTING TEMPERATURE ON THE COLOR OF OILS.
Solid line indicates red
Dashed line indicates yellow

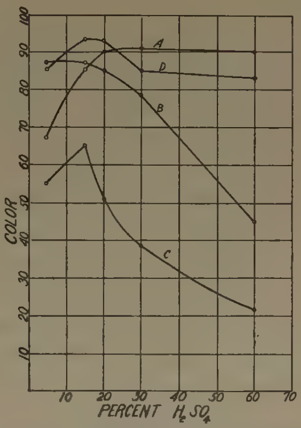

FIG. 2.—EFFECT OF DIFFERENT ACID TREATMENTS ON BLEACHING CAPACITY OF AN ADSORBENT FOR DIFFERENT OILS.

A. Texas oil
B. Oklahoma oil
C. California oil
D. Soya bean oil

The treated oils were examined for their content of red, yellow, and green colors by the Ives tint photometer, as previously mentioned. These results, showing the effect of temperature on bleaching for the colors red and yellow, are displayed in Fig. 1. The solid curves show it for red and the dotted curves for yellow. For each oil the effect for red and that for yellow are nearly parallel, so that the red colors may be used to show the trend of bleaching. Good bleaches were obtained over a considerable temperature range, and, although there was some difference in the optimum treatment temperature, satisfactory results for all the oils were obtained at 150° C. This temperature has been used for all later tests in contacting motor oils.

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[15] H. L. Kauffman: Arizona Halloysite Suitable for Clay Pulp Contact Filtration. *Ref. & Nat. Gas. Mfg.* (1927) **6**, 68; also private letters from oil-refinery chemists.

The more viscous petroleum products may require a temperature of 150° to 260° C.,[16] while lighter fractions are usually treated without heating. The vapor-phase method, in which the vapor of gasoline or other product is brought into contact with the adsorbent, is sometimes used.[17]

Preparation of Acid-treated Adsorbents

There is a decided difference between the action of acids on fuller's earths and on other earths which may be so treated as to make them efficient oil bleachers. Most fuller's earths are not changed much by the action of weak sulfuric acid solutions, but such a treatment may remove impurities from some so as to improve their grade.[18] A drastic acid treatment reduces the value of most fuller's earths, but such a treatment applied to certain earths (which may possess no value as bleaching agents in the raw state) may produce a very excellent adsorbent. Sulfuric acid is generally used (hydrochloric acid has been used in Germany)[19] and its quantity and other conditions of treatment to produce the best products have to be determined for each earth by tests.

The 17 samples of the earths previously listed were tested to determine the effect of different acid treatments on their bleaching power for the four oils. Of the several methods by which the severity of the acid treatment may be regulated, we used that of varying the concentration of the acid-water solutions, keeping the temperature, time of treatment, and earth-solution ratio constant. Samples were ground to pass 100 mesh and heated for 4 hr. at water-bath temperature (91° C.) with five different concentrations of sulfuric acid solutions, the amount of H_2SO_4 in the solutions being 5, 15, 20, 30, and 60 per cent. by weight. The solid matter was washed with water several times to remove most of the acid and soluble salts, dried at room temperature, and ground to pass 200 mesh. Three liters of acid solution was used for each kilogram of earth.

Bleaching tests were made with these 80 products and four oils, using the conditions previously outlined (4 g. adsorbent per 100 c.c. oil, 91° C. for soya bean oil, and 150° C. for the motor oils). The amount of red, yellow and green removed was determined by the Ives tint photometer.

The results, too bulky to include in this paper, show that an earth may require a certain acid treatment to make it the most efficient adsorbent for treating one oil, but that another treatment is better when used on another oil. This is shown in Fig. 2, where color is plotted against concentration of acid used to produce adsorbents from earth No. 5, a commercial earth from Southern California. For Texas motor oil the adsorbent prepared by treatment with 25 per cent. acid solution gives the lightest color, while adsorbents prepared by treatment with 15 per cent. acid solutions give the best results with California motor oil and soya bean oil. Fig. 2 also shows that a good bleach is obtained on Texas oil by using adsorbents produced over a considerable range of acid concentration, while the adsorbent for the best bleach of California oil must be prepared with close attention to its acid treatment. The curves of Fig. 3 give the effect of different acid treatments of several different earths (the numbers on the curve refer to the number of the earths previously listed) on the color of California oil; this oil was used because of its susceptibility to the effect of different acid treatments of its adsorbents. The curves show that sulfuric acid of about 15 per cent. concentration (by weight) gives the optimum bleaching effect of several earths on a California oil. Most of the published acid-treatment methods for earths designate 17 to 20 per cent. acid.[20]

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[16] H. L. Kauffman: Design and Operation of Contact Filter Plant. *Chem. &. Met. Eng.* (1927) **34,** 158. See also Footnote 15.

[17] P. W. Prutzman: Decolorizing Oils. U. S. Patent 1653735. Dec., 1927; also V. Salmi, *Op. cit.*

[18] J. T. Porter: Properties and Tests of Fuller's Earth. U. S. Geol. Survey *Bull.* 315 (1906) 272.

[19] R. Dekert: German Fuller's-earths Deposits, Preparation, and Marketing. *Eng. & Min. Jnl.* (1925) **120,** 848.

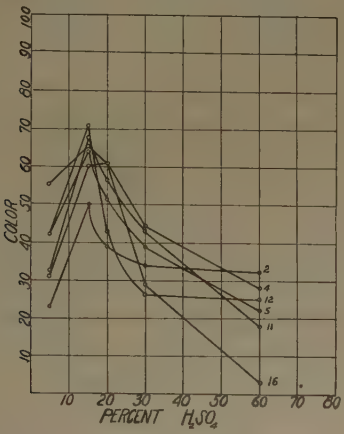

FIG. 3.—EFFECT OF DIFFERENT ACID TREATMENTS ON BLEACHING CAPACITY OF SEVERAL ADSORBENTS FOR ONE OIL

Numbers correspond to those of the list of earths, page 289.

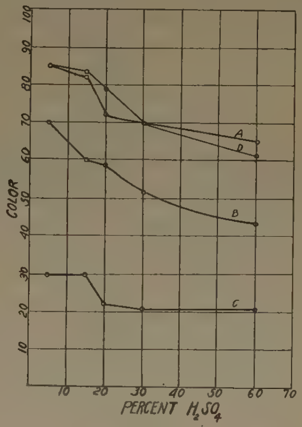

FIG. 4.—EFFECT OF DIFFERENT ACID TREATMENTS ON BLEACHING CAPACITY OF A TYPICAL FULLER'S EARTH FOR DIFFERENT OILS.

A. Texas oil
B. Oklahoma oil
C. California oil
D. Soya-bean oil

The curves of Fig. 4 are typical of fuller's earths from the southeastern United States, showing that acid treatment reduces their bleaching action; the more drastic the treatment, the greater the reduction in their decolorizing value.

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[20] Private communications from oil-refinery chemists; also V. Salmi, *Op. cit.*, and H. L. Kauffman: *Op. cit.*, Footnote 15.

BLEACHING EFFICIENCY OF ADSORBENTS

In practice, especially where petroleum products of definite colors are prepared for market, it is usual to determine the bleaching efficiency of an adsorbent by comparing the amount required to give the desired color with the amount of a product kept as a standard required to give that color. This comparison is accomplished by contacting equal samples of the oil to be treated with different amounts of the adsorbents, keeping the conditions during the treatment of both adsorbents as nearly alike as possible.

Such a method is concerned only with one resultant color. If the required color happens to be a maximum bleach, such a test is not satisfactory, for a considerable variation in the quantity of adsorbent would show little color change. Although such a comparison at some point where a small change in the amount of adsorbent shows a marked color change gives the relative efficiency for a certain color, it is not always safe to use the efficiencies thus determined as a measure of the efficiencies of these adsorbents for bleaching to some other color, as has been done in the past. However, an adsorbent with marked bleaching value at one color will probably give fair results for other degrees of bleaching on the same oil.

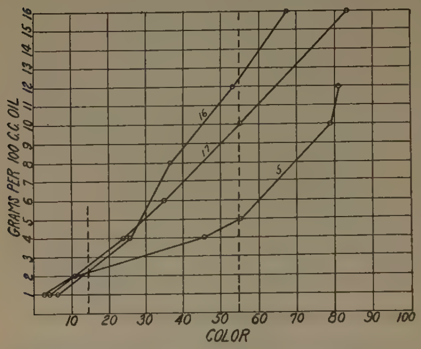

FIG. 5.—RELATIVE EFFICIENCIES OF DIFFERENT ADSORBENTS ON CALIFORNIA MOTOR OIL.

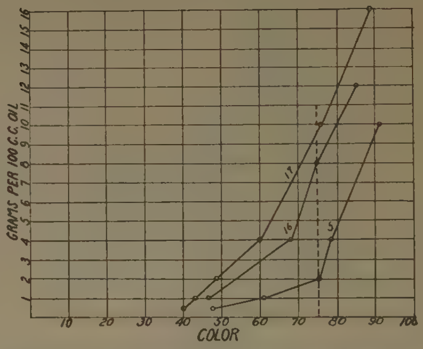

FIG. 6.—RELATIVE EFFICIENCIES OF DIFFERENT ADSORBENTS ON SOYA BEAN OIL.

Numbers correspond to those of the list of earths, page 289.

The bleaching efficiencies of the 17 earths previously listed and of their acid-treated products was examined throughout a considerable range by contacting equal quantities of the four oils used in our investigation with different amounts of the adsorbents. The color of the resultant purified oils in terms of red, yellow, and green was determined, and the colors were plotted against the grams of adsorbent used per 100 c.c. of oil. The conditions such as fineness of adsorbents, temperature, time and degree of agitation, and method of filtration were kept as nearly constant as our equipment permitted.

For the purpose of showing how the relative efficiencies of different adsorbents may change, the efficiency curves of three adsorbents (No. 5, an acid-treated commercial earth; No. 16, a Florida fuller's earth; and No. 17, a high-magnesia type of bleaching earth) for soya bean oil and for a California motor oil have been selected from the large number made. For clearness of exposition only the curves for the red are shown.

In Fig. 5, using California oil, it may be seen that for the color 14 the efficiency of adsorbents No. 5A and 16 are equal and that both are slightly better than No. 17; at 55, the color at which the oil is marketed, 5 g. of adsorbent No. 5A is equal to 10 g. of adsorbent No. 17 and to about 12½ g. of No. 16, showing that No. 5 is twice as efficient as No. 17 and two and one-half times as efficient as No. 16; at color 80 the difference is not so great, being 15/11 with respect to No. 5 and No. 17.

Fig. 6 shows the different efficiencies of the same adsorbents used with soya bean oil. At color 75 the efficiency of No. 5 is four times that of No. 16 and nearly five times that of No. 17, whereas at color 80 the efficiency of No. 5 is only twice that of No. 16 and two and one-half times that of No. 17.

The curves of Figs. 5 and 6 also show that although adsorbent No. 17 is more efficient than No. 16 for California motor oil, the reverse is true for soya bean oil. This is in accord with the well-known action of different fuller's earths, as material from one deposit was found to be superior for bleaching edible oils, and earth from another was found preferable for mineral products.[21]

EFFECT OF SOME FACTORS ON EFFICIENCY OF ACID-TREATED ADSORBENTS

Salmi[22] has found that the optimum moisture content of adsorbents depends on the product being treated and the adsorbent used, but in general is determined by the water present in the oil and the temperature at which the oil is contacted. He states that the optimum moisture content of adsorbents used with water-free edible oils contacted at about 100° C. is approximately 15 per cent. Edible oils containing water and light petroleum fractions contacted at atmospheric temperature are best treated with water-free adsorbents.

The less volatile petroleum fractions which are treated at comparatively high temperatures are usually more effectively treated by an adsorbent containing considerable moisture, and they may be treated by an adsorbent mixed with water to form a sludge.[23]

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[21] H. K. Shearer, *Op. cit.* and C. L. Parsons, *Op. cit.*

[22] V. Salmi: *Op. cit.*

[23] G. Reid: Clay as Applied as Pulp for Treating Lubricant Stock. *Ref. & Nat. Gas Mfr.* (1928) **7**, 63; see also V. Salmi, *Op. cit.* and H. L. Kauffman; *Op. cit.*, Footnote 15.

On contacting at elevated temperatures the steam liberated aids agitation, helps to prevent oxidation of the oil at the surface, and may favor adsorption of impurities by leaving an air-free surface on the adsorbent.[24]

Effect of Heat on Adsorbents

Most fuller's earths may be heated at 650° to 700° C. without destroying their efficiency, a property made use of in burning impurities from earths that have been used for the percolation of petroleum products.

To determine the action of heat on different adsorbents used in the contact process, samples of a number of adsorbents were heated at different temperatures.

The losses of water and efficiencies of the different products in removing red, yellow, and green from four oils (using 6 g. of clay per 100 c.c. of oil) were determined.

Results for typical adsorbents are given in Table 2, which shows the red colors of samples of Oklahoma oil after contacting with these products, and the loss of moisture at the different temperatures. These results show that for Oklahoma motor oil all adsorbents show a small loss of bleaching efficiency on being air dried at about 20° C. This loss is repeated as the temperature of heating is raised, until a point is reached where the bleaching power is too low for practical use. The temperature at which this occurs differs for each adsorbent.

TABLE 2.—*Effect of Temperature on Adsorbents*

Sample	Air-dried Color	110° C.—24 Hr.		235° C.—12 Hr.		450° C.—3 Hr.		600° C.—3 hr.		900° C.
		Color	Loss H_2O Per Cent.	Color	Loss H_2O Per Cent.[a]	Color	Loss H_2O Per Cent.[a]	Color	Loss H_2O Per Cent.[a]	Loss H_2O Per Cent.[a]
4A	84	79	4.92	75	2.47	68	3.93	54	0.38	4.25
6A	87	81	9.97	80	2.18	76	1.84	79	0.90	4.49
17	84	79	4.82	70	1.07	56	1.18	21	0.51	5.35

[a] The figures in this column show the additional H_2O loss caused by heating material that had already been heated at the next lower temperature.

As shown in Table 2, the high-magnesia type of earth (No. 17) had its bleaching power greatly diminished by heating at 450° and still further destroyed by heating at 600° C. This demonstrates the impossibility of heat treatment for revivifying used earths of this kind. No. 4A, a commercial acid-treated earth from near Shoshone, Calif., became inefficient at 600° C., and the bleaching power of all adsorbents tested was destroyed by heating at 900° C.

Most properly prepared acid-treated earths retain their efficiency on exposure to the air for long periods of time and are not harmed by heating for an hour at 650° to 700° C. These earths may be revivified by burning, but other adsorbents which are prepared by acid treatment and deteriorate in storage lose their efficiency rapidly when heated.[25] Heating, then, would seem to be a relatively rapid way to determine the keeping quality and to learn the possibility of revivifying acid-treated earths. Using this criterion in interpreting results given in Table 2, it would seem that No. 6A (a commercial acid-treated earth from Southern California) would stand revivifying by burning and be unaffected by storage, while the value of 4A which is destroyed on heating might be expected to deteriorate on standing. The latter supposition was not verified, because exposure for six months to the dry atmosphere at Reno had very little effect on the bleaching power of any of our acid-treated products. Storage in humid atmospheres would doubtless be more severe.

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[24] H. L. Kauffman: *Op. cit.*, Footnote 16.

Exposure to water in an attempt to restore the bleaching power of the products from the heat treatment was a failure.

The afterbleach of vegetable oils has been discussed by Parsons and others,[26] who found that this further decrease of color of bleached oils on exposure to light varied according to the fuller's earth used in the original bleach. Soya bean oil which had been bleached with some of the adsorbents used in our tests showed a slight afterbleach when exposed to bright sunlight for several days.

Samples of bleached motor oil remained unchanged in the dark, but portions of the same samples exposed to bright sunlight for two weeks became slightly darker.

SUMMARY

(1) Most fuller's earth deposits are sedimentary, but one in Arkansas is said to be the result of the alteration of basic dikes, and another occurrence in Massachusetts is a glacial silt.

(2) An examination of thin sections of a California bleaching earth of the high-magnesia type gave evidence of its being the product of the alteration of a basic crystal tuff.

(3) Additional evidence for this theory was furnished when a somewhat altered tuff from near Boca, Calif., was contacted with a motor oil and found to bleach fairly well. An examination of a thin section of this material showed it to be a basalt lithic tuff, the ground mass of which resembled the California high-magnesia bleaching earth.

(4) Slides were prepared of California earths which, when acid-treated, produce commercial adsorbents. These may be divided into two classes, those which show the texture of volcanic ash and those too fine-grained to exhibit a characteristic texture.

(5) Of several clay minerals tested, only montmorillonite when acid-treated gave a product which had good bleaching action.

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[25] V. Salmi: *Op. cit.*

[26] H. K. Shearer: *Op. cit.* and C. L. Parsons: *Op. cit.*

(6) The chemical compositions of fuller's earths and of earths from which oil-bleaching adsorbents may be prepared are widely divergent, but a high silica to alumina ratio is characteristic of them.

(7) Several tests that have been used to identify fuller's earth are only applicable to material from certain districts.

(8) The property of swelling in water seems to be peculiar to earths that become good adsorbents by acid treatment; but not all earths that swell make good oil bleachers on treatment with acid.

(9) The optimum temperature for "contact," which is determined by the character of the petroleum product being treated, was found to be nearly the same for the three motor oils used.

(10) The concentration of sulfuric acid in the water-acid solution that produced the best adsorbents ranged from 15 to 20 per cent. by weight; more concentrated solutions were detrimental.

(11) The order of the efficiencies of two adsorbents for motor oil may be different from their order for soya bean oil, and the relative efficiencies of two adsorbents used with one oil may change considerably at different points in the bleaching.

(12) The optimum moisture content of an adsorbent depends on its nature and that of the oil but is largely determined by the temperature of contacting and the quantity of moisture in the oil.

(13) Most fuller's earths and many acid-treated adsorbents are not affected by heating, but high-magnesia bleaching earth loses its bleaching value upon being heated.

(14) Soya bean oil as bleached by many adsorbents is somewhat lightened by exposure to sunlight, whereas some bleached motor oils become slightly darker under the same treatment.

ACKNOWLEDGMENTS

The authors wish to thank those of the faculty of the University of Nevada and of the staff of the U. S. Bureau of Mines who by their suggestions assisted in this work, and are especially grateful to J. C. Jones and V. P. Gianella for aid in preparing lantern slides and interpreting the thin sections. Thanks are expressed to R. H. McCarthy, under whose direction the viscosities of the oils were determined.

The authors are also indebted to the following persons and companies for the oils and earths used in the investigation and for information relative to present practice in contacting petroleum products: The Associated Oil Co., The Death Valley Clay Co., The Empire Companies, The Filtrol Co., The Marland Refining Co., The National Refining Co., The Roxana Petroleum Co., The Standard Fuller's Earth Co., The Texas Co., The U. S. National Museum, William Maddick, and M. C. and M. A. Turner.

Transverse Faults at Kennecott and Their Relation to the Main Fault Systems

By Samuel G. Lasky,* Kennecott, Alaska

(New York Meeting, February, 1929)

Faulting at Kennecott, with its attendant fracturing, is unusual, complex, and important. As study and knowledge of the various fault systems have progressed, appreciation of that importance has helped in the selection of proper horizons for exploration and in the avoidance of useless crosscutting and drifting.

During the study of this faulting and fracturing, a suggestion concerning the formation of the ore-bearing fissures occurred to me. Further study in the available portions of the approximately 45 miles of workings in the Kennecott mines for the purpose of checking up the the suggestion led to the hypothesis which this paper will attempt to present. Briefly, the hypothesis may be stated as follows: The formation of transverse faults that stop against a main system is connected with changes in the strike or dip of the main fault, and such cross faults or fractures lie in the moving block of the main fault system and indicate a change in strike or dip of the main fault.

It has been found that each kind of change tends to produce definite effects and that careful study of the evidence can often suggest the exact type of change.

OUTLINE OF KENNECOTT MINES GEOLOGY

The mines at Kennecott are on the northeast flank of a broad anticline,[1] where erosion has exposed the Upper Triassic Chitistone limestone resting conformably on the Permian or Triassic Nikolai greenstone.[2] These two are the only formations exposed in the immediate vicinity of the mines that bear relation to the ore deposits. The Nikolai greenstone is a series of altered basaltic lava flows, the actual thickness of which is unknown.

The Chitistone limestone is the ore-bearing formation. At its base is a bed of about 5 ft. of red, green, or black shale which shows strong evidence of movement wherever it is exposed underground—one of the many bedded faults of the district, caused by the slipping of the beds during folding. Above the shale lies a succession of beds; named in the order of their deposition, these are: thin bedded shaly limestone, pyritic limestone, a crinkly appearing limestone, a dull gray or black limestone, and a glistening limestone in thicker beds. Any of these beds may be absent at any section. The total thickness of this succession is 90 to 100 ft., though locally it has been found as thick as 130 ft. and as thin as 60 ft. Above this is a very thick-bedded dolomitic limestone. The dolomite is not an original deposition confined to certain beds, although the average strike of the limestone-dolomite contact roughly approximates the strike of the beds, grading sometimes sharply, some-

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* Kennecott Copper Corporation.

[1] A. M. Bateman and D. H. McLaughlin: Geology of the Ore Deposits of Kennecott, Alaska. *Econ. Geol.* (1920) **15,** 10.

[2] F. H. Moffit and S. R. Capps: Geology and Mineral Resources of the Nizina District, Alaska. U. S. Geol. Survey *Bull.* 448 (1911).

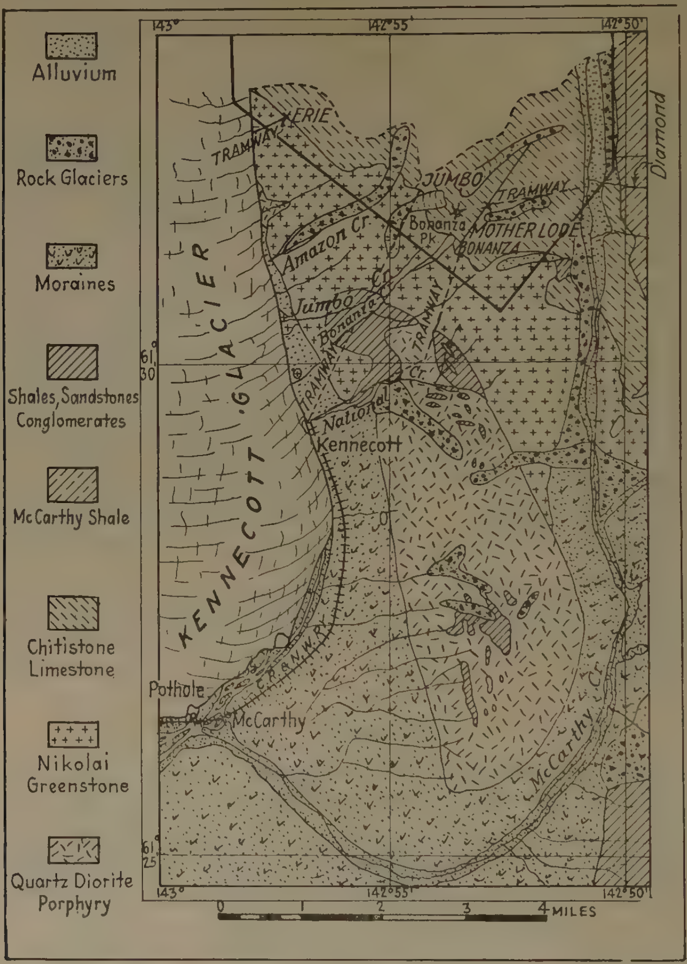

Fig. 1.—Geological map, vicinity of Kennecott (from U. S. G. S. Bull. 662).

times insensibly, from the dolomitic limestone, which contains about 16 per cent. MgO, to the limestone of the lower beds, which contains less than 1 per cent. Elsewhere, however, the dolomite may stop abruptly at either a bedding plane or a thin gouge seam. Visual distinction between the limestone and the dolomite is usually easy.

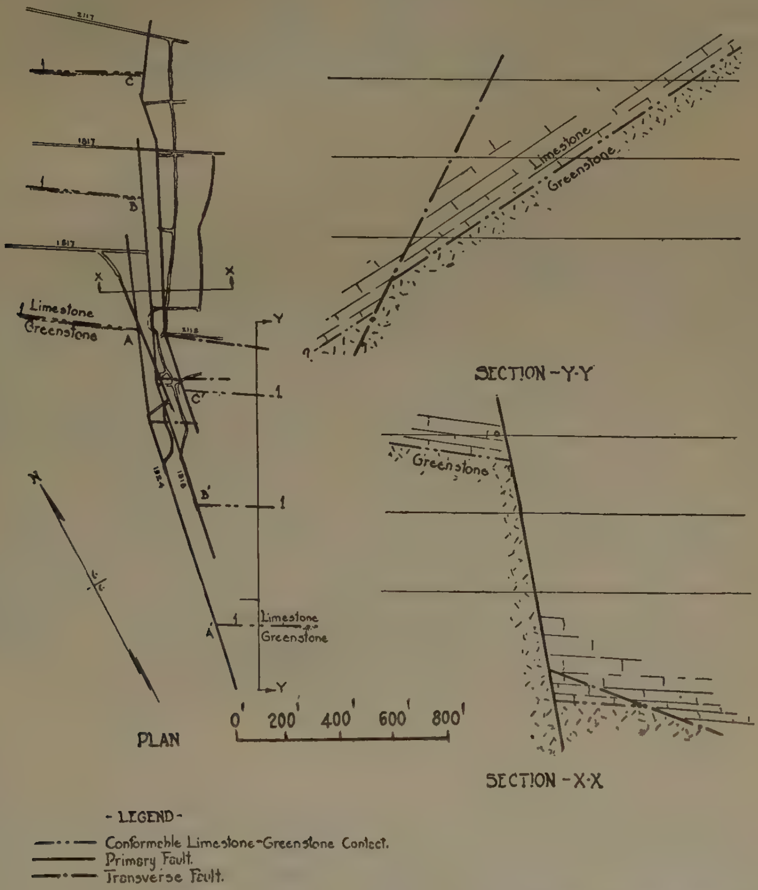

FIG. 2.—PLAN OF PORTION OF WORKINGS AND SECTIONS SHOWING PRIMARY AND TRANSVERSE FAULTS EXPOSED ON THREE LEVELS.

The average strike of the bedding is about N. 60° W., and the dip from 22° to 40° N. E., steepening to the northwest and also down the dip (Fig. 2). Several small transverse folds lie on the flank of the main fold, with their axes paralleling the dip of the beds and pitching with them. They cannot be seen on the surface but have been revealed underground in the course of mining. These transverse folds, forming grooves and ridges down the dip of the beds, are important.

The copper ore occurs in the Chitistone formation and seems to favor the dolomitic phase at or near its contact with the limestone. For this reason, preliminary exploration is restricted to that horizon of the dolomite. That mineralization favors the dolomitic phase may be due to the greater fracturing in the brittle dolomite, rather than to preferential replacement, since fissures in an ore zone extending into the limestone from the dolomite are equally mineralized throughout. The location of the ore depends upon the location of the fracturing which, in turn, is controlled by the bedded faults, as I will attempt to show.

Faulting

Three main types of fracturing are exposed in the Chitistone limestone; faults which extend into the greenstone and displace the entire formation, bedded faults, and the zones of fissuring in which the ore mainly occurs. Most of the faults displacing the full formation are premineral; some have also postmineral movement. A few of them have, in a local sense, large displacement; there are many of small displacement. The stratigraphic displacement of the limestone-greenstone contact is the only common evidence available for the solution of these faults, and it is difficult to determine how much of that displacement is due to strike movement and how much to dip movement.

All of the major faults are normal with the exception of one. In general they strike about N. 20° E., dip 65° to 90° SE., and are accompanied by parallel zones of fissuring and fracturing and by bedding-plane slips and other evidences of disturbance transverse to them. The transverse structures, together with the fissuring in which the ore occurs, form the basis of this paper and will be discussed fully later.

Bedded Faults

Bedded faults are numerous. They are low-dipping planes of movement which coincide with the bedding or cut across it at very slight angles. Some are transverse structures related to the large steep faults, as mentioned, but most are the result of the slipping of the beds during the folding of the major anticline. There are joints roughly parallel to the strike of the beds which dip 60° to 75° SW. Applying the theory of the strain rhomb, the upper beds are found to have moved relatively up the dip and the lower beds down, as would be expected. The bedded faults are then thrust faults with the movement up the dip of the limb of the main fold; they are approximately dip-slip faults with respect to the main fold, and strike-slip faults with respect to the minor flexures on its flank, whose axes approximately parallel the dip of the flank. If those are the movements that actually have taken place, any fractures along the minor flexures were not formed altogether in the usual manner of formation of fractures along the axes of folds, in which case the slippage along the bedding planes is up or down the flank rather than along it. It is for this reason that the minor folding on the flank of the major fold at Kennecott is unusual; the axes of the minor folds are not parallel to the axis of the major fold but are approximately parallel to the dip of its flank, so that the movement of the beds is approximately *along* the flanks of the minor folds, as has just been pointed out, and not up or down them.

Mineral-bearing Fissures

The veins have been formed by the filling of fissures and by replacement along zones of fracture.[3] Both the shape and location of the mineral-bearing fissures and fracture zones, and of others of the same type which are not mineral-bearing, are closely connected with the bedded faults through the transverse minor folding. It has been found that the fracturing associated with mineralization is localized along the crests of anticlines and the bottoms of troughs caused by the light transverse folding on the major flank. It has been suggested that this fracturing was not formed in the usual manner of fractures along the axes of folds; this paper attempts to give a different explanation and to show the similarity of the fractures to those transverse to the steeply dipping northeast faults. It is the contention here that the fractures are transverse structures dependent on the faults and are controlled by changes in strike, and perhaps dip, of the faults.

In shape, the fissures and veins are roughly long thin wedges, usually in the hanging wall of a bedded fault, with the base of the wedge resting upon the fault. The longest dimension of the wedge is the length of the base, which is many times the distance from base to apex; likewise, the distance from the base to apex is many times the width of the base. The fissures narrow rapidly out and away from the bedding-plane faults; the frayed upper edge of the fissure, being the apex or sharp edge of the wedge, pitches roughly parallel to the base. In some cases the wedge is inverted; that is, the fissure is topped by a bedded-plane fault and lies wholly within the footwall of the fault (Fig. 3). One of the large stopes shows this condition, the ore in the back terminating abruptly against an overlying bed fault. Instead of the fracture of the rock resulting in a definite fissure, it may occur only as a zone of crushing or shattering, but this crushed or shattered zone tapers in wedge form. Whether crushing or fissuring results would depend on whether the acting force were pressure or tension. In a few instances the apex of the wedge starts from a bedding-plane fault and widens away from it, probably to stop against an overlying plane of movement.

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[3] A. M. Bateman and D. H. McLaughlin: *Op. cit.*, 21, 26. A. M. Bateman: Angular Inclusions and Replacement Deposits. *Econ. Geol.* (1924) **19**, 513.

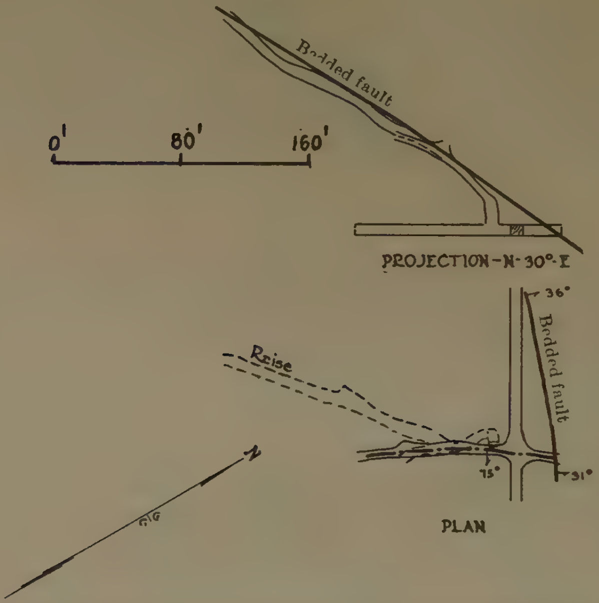

Fig. 3.—Example of fissure in footwall of bedded fault.

Evidence in Favor of the Hypothesis

Bedded Faults

Fig. 4 is part of a geologic map of one of the levels and shows evidence as to the location and shape of the fissuring and fracturing controlled by the bedded faults. It shows also how the minor folds were, in part, determined. There are three prominent changes in the strike of the bedded fault; that is, there is a minor anticline with a broad crest and the changes in strike occur at each edge of the crest and at the foot of the western limb, which is probably part of an adjacent syncline. At each of these points of changing strike there has been some type of fracturing transverse to the bedded fault. In the two cases where drifts follow the transverse fractures they were found to stop against the bedded fault. At *C*, the transverse fracture has taken the form of a strongly broken and crushed zone with a wide base at the bedding-plane fault, tapering gradually away to a narrow crushed zone. At *B*, the flexure seems slighter and the transverse structure is a calcite vein of uniform size, so far as followed. At *A*, the cross faulting was partly explored in only one direction; it is a broad zone of fracturing along which there has been some movement. Admittedly, it may or may not stop at the bedded fault. The limbs of the minor folds show different dips. Fig. 5 is a diagrammatic representation of these conditions. The transverse fractures are lettered as in Fig. 4.

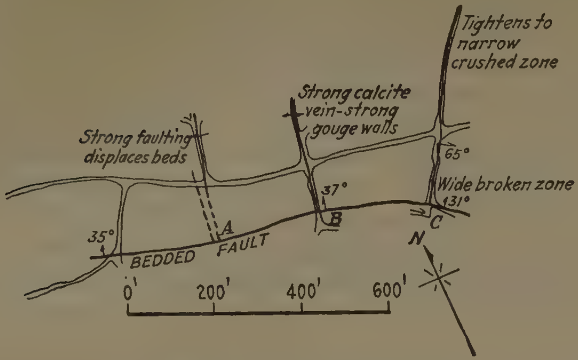
Tightens to narrow crushed zone
Strong calcite vein-strong gouge walls
Strong faulting displaces beds
65°
37°
Wide broken zone
31°
35°
A
B
C
N
BEDDED FAULT
0'
200'
400'
600'

FIG. 4.—PART OF GEOLOGIC LEVEL MAP SHOWING PROBABLE BEDDED THRUST FAULT WITH TRANSVERSE FRACTURES.

These are type examples. Where the flexure covers a greater area and is more gradual, the transverse fracturing is more likely to be distributed in zones than to be concentrated in a single fracture. Fig. 3 illustrates a transverse fissure in the footwall rather than the hanging wall of a bedded fault.

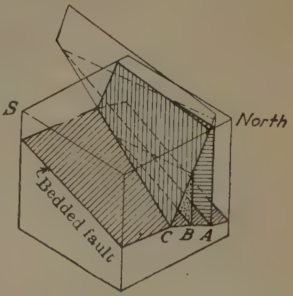

FIG. 5.—ILLUSTRATING CONVERGENCE OF AXES. A VIEW OF CONDITIONS SHOWN IN FIG. 4.

The Steep Northeast Faults

It has been mentioned that the major faults of the Kennecott district are accompanied by much transverse disturbance. All such disturbances seem to lie within a single wall of the fault and, with the exception of the few transverse bedding plane slips, are in most respects similar to the fractures transverse to the bedded faults. Not only do they form similar types of fracture, but they also occur, it seems, at points of changing strike or dip, or both, along the faults. Such a similarity is significant; it more than hints that the kind of actions that produced the fracturing transverse to the many bedded faults also caused the fracturing transverse to the steep northeast faults. In both cases the transverse fracture may or may not show evidence of movement, though such evidence is more frequent in the fractures associated with the steep northeast faults. It is only in shape that the transverse fractures along bedded faults differ from those transverse to the steep faults; they are more often wedge-shaped along the bedded faults.

Fig. 6 shows the structures exposed on one level in the hanging wall of a strong fault. Although only one of the transverse fractures has been explored to the fault, from the comparative strength of the fractures and the fault, which is a very strong one, it seems reasonable to assume that the fractures do stop at the fault. The position and strike of the fault as found at *A* and *D*, and the location of the face of the crosscut at *B*, indicate a decided synclinal warp in the fault plane. Curiously enough, the extension of the fault from *A* and the extension of many of the fractures, intersect at almost a common point. If then, the strike of the fault is slightly changed at this point and at each succeeding intersection of fracture and fault, in a direction suggested by the type of fracture, the strike and position of the fault on reaching *D* coincides closely with the actual condition of the fault as found there. Unfortunately, this warp in the fault plane has not been explored on any other level.

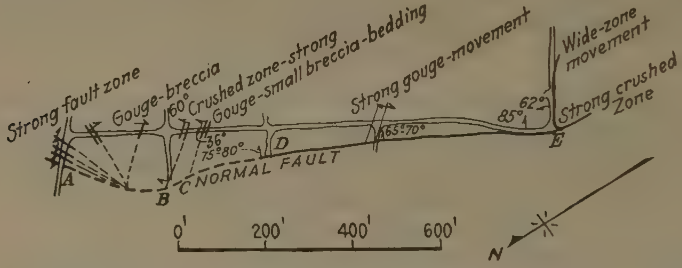

FIG. 6.—PART OF GEOLOGIC LEVEL MAP SHOWING STEEP NORMAL FAULT WITH TRANSVERSE FRACTURES.

However, the transverse fracture indicated at *E*, and the accompanying fault flexure, have been explored on other levels besides the level shown. Here there is no assumption or inference but an actual condition which may be studied. The transverse disturbance starts at a point of synclinal flexure in the fault plane as a zone of strong crushing and opens to a wide zone of many slickensided slips. It does not cross the fault. Fig. 2 shows the relation of this same transverse disturbance and the main faults of Fig. 6 as exposed on three levels, a vertical extent of 600 feet.

At *C* is apparently an instance of a bedded fault as a transverse coordinate structure of a major steep fault.

Transverse Fracturing in the Moving Block Only

In both the faults cited, the transverse fracturing has been wholly in one wall of the fault. Such is the general condition of all faults throughout the property. Although Fig. 6 shows no exploration in the footwall of the main fault, the footwall has been explored on other levels. Since the major stresses are likely to be set up in the moving block of a fault, it is in this block that the transverse fractures will be expected. Movement probably takes place, in some cases, in both blocks, and a transverse fault in each block may meet at the main fault and appear as a single slip crossing the fault, but that would be a rare coincidence. Though movement may take place in both blocks, most of the movement may be expected in one block of the fault, and where much transverse disturbance is found in one wall of a strong fault, and practically none in the other wall, it is reasonable to assume, in the absence of data to the contrary, that that is the wall of the moving block. If this is true, the movement of the fault shown in Figs. 2 and 6 has been almost wholly in the hanging wall; in Fig. 3, almost wholly in the footwall.

General Discussion

Assuming that the explanations given are correct; that the bedded faults are thrust faults, and that the moving block is indicated by the transverse fracturing, the major fracturing shown in the sketches may be tabulated thus:

Fig. 3. Thrust fault, footwall movement, anticlinal flexure. Transverse fissure is slightly stronger at fault (Fig. 7).

Fig. 4, at *C*. Thrust fault, hanging-wall movement, anticlinal flexure. Wedge-shaped broken and crushed zone tightening away from fault (Fig. 8).

Fig. 6, at *E*, and Fig. 2. Normal fault, hanging-wall movement, synclinal flexure. Crushed zone showing evidence of movement (Fig. 9).

To illustrate these conditions of faulting and fracturing, sketches were made (Figs. 7, 8, 9). Each condition of faulting seems to have its own type of cross fracturing; either a compressive or tensile action, resulting respectively in crushing or fissuring. The transverse fracture zone, especially when not associated with bedded faults, may be either tabular or wedge shaped, and with or without movement. In the cases where the transverse fracture zone is wedge shaped, the location of the base of the wedge is indicative of the kind of movement. With crushing, local faulting may take place along the transverse zone in the squeezing out of small wedges, or there might be displacement along the two walls. With two exceptions, the fractures transverse to either steep or bedded faults are the same. One exception is the comparatively flat dip of transverse bedded faults (Fig. 6, *C*). All other transverse fractures are fairly steep. The other exception lies in the fact that fractures transverse to other than bedded faults are not so likely to be wedge shaped.

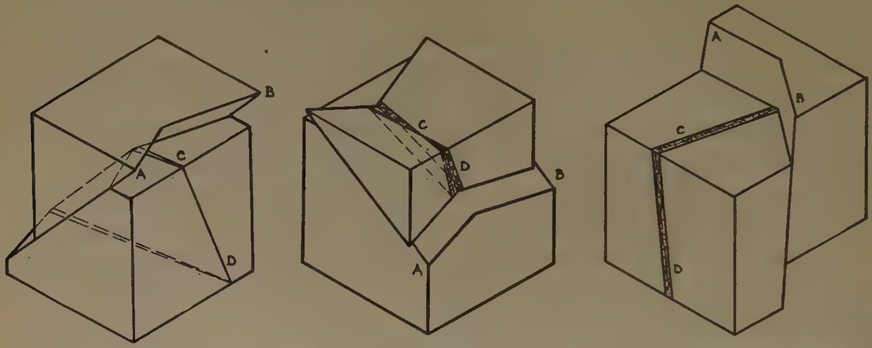

FIG. 7.—Footwall movement.

FIG. 8.—Hanging wall movement.

FIG. 9.—Hanging wall movement.

FIGS. 7–9.—CONDITIONS OF FAULTING AND FRACTURING. *A-B* = MAIN FAULT; *C-D* = TRANSVERSE FRACTURE.

FORMATION OF TRANSVERSE FRACTURE SYSTEMS

In opposition to the hypothesis, the argument may be made here that where ridges and grooves exist along a fault surface resulting in a change in strike along the plane, the movement of the faulted blocks may be expected to take place in the direction of the axes of the grooves and ridges, in which case there might not be the transverse fracturing as suggested by the hypothesis. That can be true only if the axes are parallel. With only the slightest divergence of the axes from the parallel (Fig. 5) transverse fracturing must be formed in brittle rock such as the dolomite of Kennecott, each groove or ridge offering an opposing factor to unimpeded movement along the adjacent groove or ridge, with consequent fracturing in the moving block to relieve the stress. The stress will be greatest at the crests and troughs, and there relief will take place. The movement along any flexure will have to be parallel with the movement of the fault, of course, but will tend to be up or down the dip of each limb. In overcoming this tendency and the opposition of adjacent flexures, either a tension or a compression along the axis will result, with fissuring when due to tension and crushing when due to compression. Whether the force is tension or pressure will depend on whether the flexure is a groove or a ridge, whether the movement is normal or reverse, and whether the movement is in the footwall or the hanging wall. The widest part of the transverse fracture zone that will be formed—*i.e.*. the base of the wedge—will be at the region of greatest stress.

A study of the figures and an example will make the foregoing clear. Take the case of Fig. 8, an anticlinal flexure along a thrust fault with hanging-wall movement. Although the different actions of folding, bed movement and fracturing take place simultaneously, for the purpose of explanation they may be ascribed relative ages. As a fold in brittle rock takes place, wedge-shaped fissures or zones of fissuring would be formed along the axial zone of the fold, in some such plane as *CD* of Fig. 8. The widest part of each single fissure would be in the beds farthest from the theoretical center of curvature of the fold, where the stress is greatest, with the fissure narrowing toward the center of curvature, to finally approach a width of zero at the line in the plane where the tension was no longer great enough to rupture the rock. The fissure, then, would separate the fold into two blocks, which would be in contact with each other along the line where the width of the fissure is zero. With movement along the fault *AB*, the movement of the blocks to the right and left of *CD* will tend to be up the right and left flanks of the flexure respectively, each block opposing the other. Crushing should take place, beginning along the line of contact of the two blocks, where the width of the original fissure is zero; as the movement continued, the result should be a gradual closing of the fissure accompanied by crushing of the walls of the fissure as they come into contact with each other. The final structure would be a wedge-shaped crushed zone in place of the original wedge-shaped fissure, with the difference that the location of the apex and base of the wedge is reversed. If the fault *AB* were not a bedded fault, in which case no initial fissure due to bending would be formed, or if the flexure of the bedding was not great enough to cause initial fissuring, the pressure would be evenly distributed over a plane and the resulting crushed zone probably would not be wedge-shaped. If the actions described above are true, they offer the reasons why transverse fracture zones associated with the movement of the steep northeast faults of Kennecott are not usually wedge-shaped, while fractures transverse to bedded faults usually are so.

Using this line of reasoning, the kind of transverse fracture that would be inferred from any individual fault flexure at Kennecott seems to check reasonably what is actually found.

Relation between Flexure and Strike and Dip of Transverse Fracture

As to the strike and dip of the plane *CD*, certain relations became evident at the beginning of the study. These relations are embodied in what I have called The Rule of Strike Relations of Fault Flexures, which may be stated as follows: When viewed down the dip of a fault, the bearing of the axis or line of intersection of the limbs of a fault flexure makes the smaller angle with the strike of the flatter limb when the flexure is anticlinal, and with the strike of the steeper limb when the flexure is synclinal; and any transverse fracture having the axis of the flexure as its base or apex will always dip toward the bearing of that axis (Fig. 10).

The line of intersection of the limbs of a fault flexure is the axis of the flexure, and the transverse fracture formed will have that axis as its base or apex. The transverse fracture may take any strike or dip, from perpendicular with an intermediate strike to a dip and strike equaling that of either limb—just as the leaves of a new book may fan out when the book is partly opened and tipped—but with any strike under the above conditions there can be only one dip, and with any dip only one strike. If the limbs of the flexure have equal dip, the bearing of the axis will be equidistant from the bearings of the limbs; but if the steepness of the limbs is unequal, the axis swings to one side or the other of the median line, as stated in the rule and as illustrated in Fig. 10.

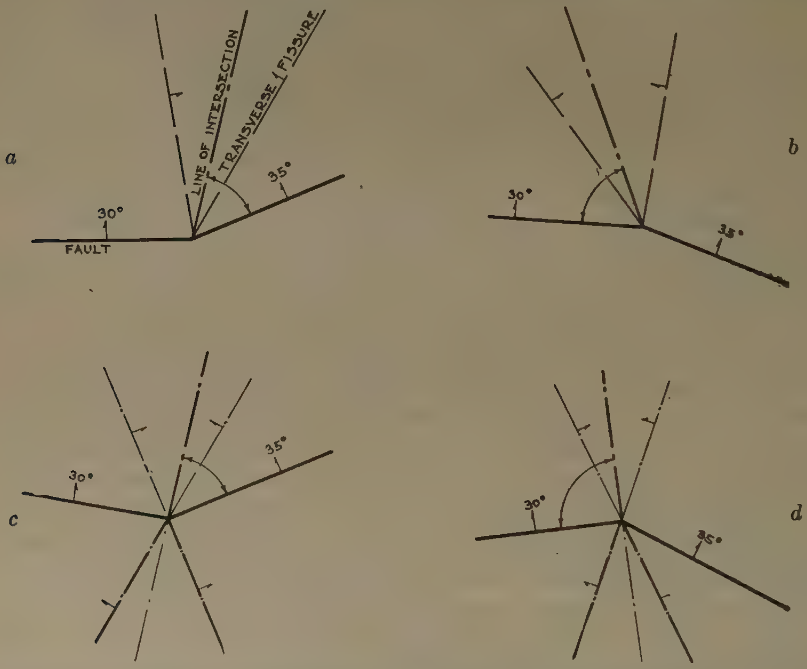

FIG. 10.—PLAN VIEWS ILLUSTRATING THE RULE OF STRIKE RELATIONS OF FAULT FLEXURES.

In any faulting with which there is associated transverse fracturing there are at all times five variables:

1. The strike of the transverse fracture,
2. The dip of the transverse fracture,
3. The relative dip of the limbs of the flexure,
4. The relative strike of the limbs of the flexure,
5. The strike of the axis of the flexure.

Given any three, the others may be predicted. Let us take as an example the conditions shown in Fig. 4 at *C*. A transverse fracture is shown at an anticlinal flexure of a bedded fault; a definite strike and dip are exposed. Three variables, then, are given—1, 2, and 4. The strike and dip of the transverse fracture gives a definite strike to the axis of the flexure, which axis makes the smaller angle with the strike of the eastern limb of the flexure. From the suggested rule, the eastern limb should be the flatter, and the dip that is actually exposed checks that conclusion. Assuming, however, for the sake of illustration, that exploration work had not advanced far enough to expose the strike or dip of the limbs, the conclusion would be that there exists either an anticline with a flatter eastern limb or a syncline with a steeper eastern limb. At least we would know that the fault could not continue with the same strike or dip, else there would be no transverse fracture. An examination of the transverse fracture itself should tell what is the actual condition. The transverse fracture is present in the hanging wall of a bedded thrust fault as a crushed zone with the base of the wedge at the fault. Such a condition is associated only with an anticlinal flexure and in any further exploration the fault should be looked for in the location suggested by an anticlinial flexure with the eastern limb flatter than the other.

Application of the Major Hypothesis

Following are a few examples of the application of the theory of this paper.

1. The reference plane for the approximation of the correct horizon for preliminary exploration—the limestone-dolomite contact—is the greenstone-limestone contact. The thickness of the limestone between the greenstone and the dolomite is determined, usually by diamond-drilling, in each locality and that thickness assumed as constant throughout the locality. Wherever the greenstone-limestone contact is found on any level displaced by faulting, exploration crosscuts are driven to conform to the displacement. Fracturing transverse to bedded faults would not disturb the contact, but that transverse to the steep northeast faults might do so (See Fig. 2).

2. Since the limestone-dolomite contact roughly follows the bedding, a change in the strike of a bedded fault would suggest a possible corresponding swing in the limestone-dolomite contact; therefore a study and correct analysis of the fractures exposed in an exploratory crosscut would insure that the crosscut would remain more nearly in the correct horizon. Unless the flexure were very strong, however, this application might be modified by operating requirements.

3. Another application has been given in the analysis of the conditions at *C* of Fig. 4.

4. The crosscut at *B* of Fig. 6 was driven to cut the fault. After having been driven much farther than had been expected, it was finally stopped. An examination and correct analysis of the fracturing exposed in the drift could have indicated fairly closely the position of the fault. Had it been known how far the crosscut would have to be driven, probably it would not have been run until the drift had passed the curve in the fault.

5. In one of the mines there are two mineral-bearing faults roughly parallel with each other, which, on the level to be described, are approximately 350 ft. apart. They were explored by a drift along fault *A* and crosscuts from the drift to fault *B* at intervals of about 225 ft. Ore was struck along fault *A* at a point of strong flexure but abruptly turned off into the transverse fracture that is associated with the flexure. However, the transverse fracture dissipated into a wide zone before reaching the other fault and the ore did not continue across. Its total exposure was about 50 ft. along fault *A* and slightly greater along the transverse zone. Had the exploration been by a drift along fault *B*, instead of along fault *A*, an alternative that was considered, and crosscuts driven to *A*, the ore would have been found only by accident; it would not have been cut by the crosscuts, since its exposure along the fault is but a fraction of the crosscut interval; nor would it have been cut by the drift along fault *B*, since it extends only a part of the distance between the two faults. And since orebodies of that type are very unusual at Kennecott, its existence would never have been suspected. However, the crosscuts, though they might not have cut the ore, would have indicated by the position in which they intersected fault *A* that it had made a decided turn. An application of the theory would immediately suggest a zone of fracturing at the flexure; which, coupled with the fact that it was transverse to a mineral-bearing fault, would have led to investigation and disclosure of the ore.

6. A solution has been advanced for one of the fault systems at Kennecott which has as a necessary component the assumption that only the hanging wall of a certain fault had moved. The theory of transverse fracturing applied to that fault seems to uphold the assumption, but regardless of whether or not the solution of the system is correct, it illustrates the application of the theory to problems of that kind. If the theory and its application in this special case are correct, the assumption that movement was mostly in the hanging wall of the fault is no longer an assumption but becomes a fact, and the solution of the fault system is advanced by that much.

SUMMARY

It has been found that transverse disturbances exist at certain definite points along all types of faults at Kennecott—namely, at flexures in the fault surfaces; that they occur seemingly in but one wall of the fault; that they are not all of the same type or form; and that different forms and different shapes occur with different types of fault movements.

As a result of a study of these conditions, the hypothesis is presented that transverse disturbances found stopping against a main fault are dependent on the presence of flexures along the fault surface; that they occur in the moving block and indicate that block by their presence: that the transverse fracture is often indicative of the kind of flexure and movement that caused it; and that a certain definite relationship of strike and dip exists between the transverse fracture and the fault flexure.

Examples of the application of the hypothesis to Kennecott geology have been given. The suggestion is made that the hypothesis may have a wider application, and it is hoped that others may attempt to test the theory by applying it in districts with which they are familiar.

Extension of Oreshoots with Comments on the Art of Ore Finding

By Harrison Schmitt,* Hanover, N. M.

(New York Meeting, February, 1929)

In the practice of ore finding the geologist is continually confronted with the question of oreshoot extension, so that the general problem seems worthy of systematic investigation. This problem appears when the reserves are estimated and when the advisability of deepening the shaft comes up. Moreover, there is need for lower development cost to keep pace with cost reductions in other departments, or even to offset a rising cost of ore finding or ore following. This can be gained by close geologic control of the development; and much of the geology involved is that bearing on the downward, upward, and horizontal extensions of shoots.

Furthermore, the purchase of a mine or prospect often depends directly on the question of shoot extension; because it is seldom that the ore is fully developed and the purchase price is almost invariably based largely on "possible" and geologically prospective ore. Most current methods for estimating "possible" and the more nebulous classes of ore are crude and indecisive.

In the following pages the extension of oreshoots is the major theme, although the more basic subject, the general art of ore finding, is given some space.

(*A*) DIRECT METHODS OF EXTENSION PREDICTION

(*I*) *Extrapolation and Interpolation of Shoots by Use of Conventional Geologic and Assay Plans, Projections and Cross-sections*

There is little need for stressing the desirability of detailed geologic plans, assay plans, projections, and cross-sections as a basis for the study and extension prediction of shoots. Underground and surface mapping, in most cases, ought to be done in exhaustive detail. All features of obvious bearing should be mapped and it is good practice to record all possible details, some of which may become significant when studied in the ensemble. It should be emphasized that the mapping should be done with an impartial mind, with interpretation reduced to the minimum. The symbols of the map can be made to distinguish observed facts from interpretation.

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Impartial, detailed mapping often automatically solves many geologic problems through the completeness of the picture presented. Much less interpretation than is commonly used, is necessary; in fact, in many cases incorrect conclusions are caused by premature judgement that clouds the fact-gathering. A good deal of cold-bloodedness is essential in mapping, for it is surprising how easy it is in geology to find facts to fit almost any opinion.

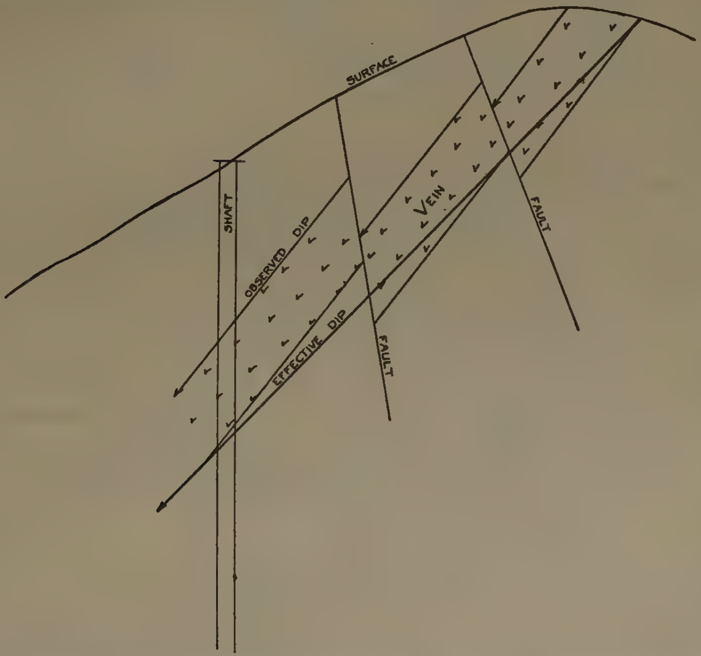

FIG. 1.—IDEALIZED CROSS-SECTION THROUGH ALFAREÑA VEIN AND ORESHOOT AT SANTA BARBARA, CHIHUAHUA; SHOWING MANNER BY WHICH FAULTING CAUSED EFFECTIVE DIP TO DIFFER FROM OBSERVED DIP.

For greatest utility in guiding development by the prediction of extension the geologic detail should be caught up at the headings at short intervals; once a week may not be too frequent during an intensive development campaign.

To get the greatest benefit from the detail much thought and care must be spent in devising a system of maps for recording it. No maps will give an entirely satisfactory three-dimension mental picture, and one can only strive to make them approach this ideal. It is seldom possible, therefore, to have too many cross-sections, projections, and structural contour maps. In many cases it is inadvisable to place the cross-sections at arbitrary intervals. It is good practice to locate them where they will "cut the most information," thus giving the best basis for oreshoot extrapolation.

To get best results on cross-sections, especially where there are low dipping contacts or veins, a definite horizontal plane for mapping on the mine level should be selected and adhered to. This plane may pass through the bottom, top, or middle of the drift. Because it is the most accessible, I prefer to map on the last.

In projecting dips on cross-sections the recorded dip may not be the effective dip of the oreshoot or vein because of distributive faulting or slicing. This was the case at the Alfareña mine at Santa Barbara, Chihuahua (Fig. 1). At Bisbee, Arizona, for the same reason observational dips do not always correspond with the composite dip of the horizon measured.

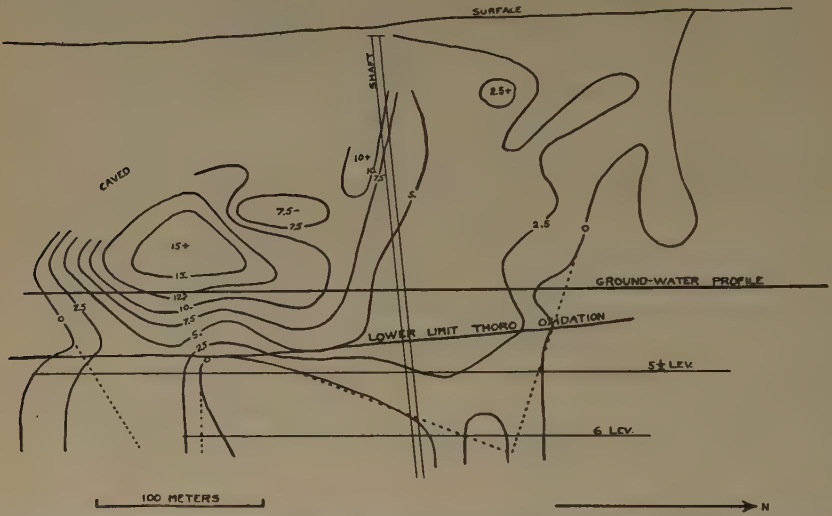

FIG. 2.—EQUAL-ORE-WIDTH LINES ON LONGITUDINAL-VERTICAL-PROJECTION OF AN ORESHOOT ON VETA COLORADO (RED VEIN) NEAR PARRAL, CHIHUAHUA. THE INTERVAL IS 2.5 METERS.

(II) *Extrapolation by the Analysis and Projection of More Detailed Information*

(*a*) *Equal-ore-width Lines.*—Equal-ore-width lines are lines connecting "points" of equal vein width (ore width) on the projection of a shoot. This gives an approach to a three-dimension picture of the shoot. The "iso" lines may be shown on a horizontal, inclined, or vertical projection, depending on the amount of inclination of the vein.

On the Veta Colorado (Red Vein) near Parral, Chihuahua, equal-ore-width lines, projected to a longitudinal vertical plane, were found useful. The known parts of the shoots appeared as a three-dimension picture and their extension downward and horizontally could be predicted with some confidence, because the border or "zero" lines are "reenforced" by the interior lines. Fig. 2 shows a shoot outlined previous to the final development. Below the 5½ level, the predicted extension of the shoot is shown in dotted line and the extension actually developed is shown in solid line.

(*b*) *Equal-ore-value Areas.*—The erratic distribution of the metal content of most shoots makes difficult the direct drawing of equal-metal-value lines. Equal-metal-value areas, however, can be outlined and these give, indirectly, equal-ore-value lines.

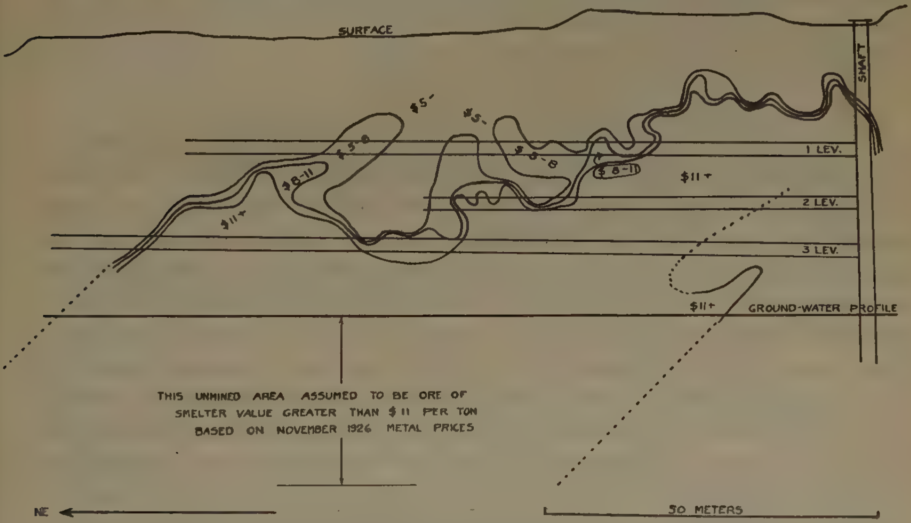
SURFACE
SHAFT
$5-
$5-
$5-8
$5-8
$8-11
$8-11
$11+
$11+
$11+
1 LEV.
2 LEV.
3 LEV.
GROUND-WATER PROFILE
THIS UNMINED AREA ASSUMED TO BE ORE OF SMELTER VALUE GREATER THAN $11 PER TON BASED ON NOVEMBER 1926 METAL PRICES
NE
50 METERS

FIG. 3.—EQUAL-ORE-VALUE AREAS AND CONSEQUENT EQUAL-ORE-VALUE LINES ON AN ORESHOOT IN CORDERO (SAN JUAN) DISTRICT, SOUTHERN CHIHUAHUA. THE INTERVAL IS $3 U. S. CURRENCY PER METRIC TON.

At Cordero (San Juan), a mining camp about 30 miles north of Parral, Chihuahua, it was found possible to study the developed, and even mined, parts of the shoots by blocking out areas of equal smelter value, this value being an approximate measure of the intensity of the mineralization. The analysis of the known parts of the shoots gave a basis for their extrapolation in depth, the interior equal-value-lines "reenforcing" the border or "zero" lines. (Fig. 3.)

This method has as its basis thorough sampling. The areas of equal smelter value are then outlined by reducing the assays of each sample to their smelter value per ton. Lead and silver were the most important metals and bore a remarkably constant ratio, one to the other, so together they made a good basis for this value classification. The proportions of gold, zinc and copper were erratic, but these metals, being of minor importance, could be disregarded.

The Cordero shoots are rather regular in outline, as shown by the projection of one of them (Fig. 3), and so an intelligent guess as to the quantity and quality of the ore in the unexplored extension downward could be made.

(*c*) *Curves Showing Variation of Metal Content.*—At the Hidalgo mine, Santa Barbara, Chihuahua,[1] exhaustive stope sampling, well recorded, made it possible to construct curves showing the variation of the metal content from near the outcrop to the lowest developed horizon. An analysis of these curves, making allowance for enrichment in the oxidized zone because of leaching of valueless material, indicated a definite trend of values in depth, an increase in this case. The extension of the shoot in depth could be predicted with some assurance.

(*B*) INDIRECT METHODS OF EXTENSION PREDICTION

(*I*) *Study of the Relation of Structural Features to the Orebody*

The study of the relation of the structure of the country rock to the oreshoot is one of the first tasks to be undertaken before attempting to predict extension. There is a growing appreciation, not yet fully expressed, of the importance of the association of oreshoots with structural and purely mechanical features of the country rock. In recent years some schools of geology have taught that mineralization tends to avoid strong faults, the theory being that gouge tended to seal them off, preventing circulation and thus deposition. There are a large number of deposits for which this does not hold. For example, in the Parral district, Southern Chihuahua, the strongest veins are in the strongest faults. There in places the mineralization seems to have replaced the fault gouge with silica and ore. Today the important water courses are along the vein-ore-faults, and one can readily imagine that such was also the case during mineralization.

The Veta Colorado in the same district is a fine example of the manner in which orebodies favor structural features. This rich lode is a vein-fault, having a dip slip of 1450 ft. at its richest portion. The best ore is directly associated with the shear plane of the fault zone that has the greatest offset. In further detail, the shoots appear to have originated by the replacement of breccia that filled depressions in the foot-wall of the major shear plane.

Examples of ore in or near strong faults, or structural disturbances of equal importance, are common. Ore recently found at Tintic is associated with conspicuous structural abnormality. At Hanover, New Mexico, the zinc ore is associated with broken "open ground," that is, almost invariably in or near the major faults. There it is felt that if the location of broken ground of premineral age can be predicted, a favorable chance for ore can be assumed; and ore has been found on this basis.

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[1] H. Schmitt: Geologic Notes on the Santa Barbara Area in the Parral District of Chihuahua, Mexico, *Eng. & Min. Jnl.* (1928) **126,** 410.

The prediction of the location of "open ground" usually requires exhaustive preliminary measurements of structure. Very few geologists, and fewer miners, appreciate the possibilities for ore finding where contacts and other details of structure are measured to the foot, and mapped on an appropriate scale. I feel that the time has come when exceptional care in ore geology mapping is in order in many mineral districts. Much new ore can and will be found by such detailed technique, as has been applied at Butte and the Homestake and in a few other districts.

(II) Study of the Regional Habits of Oreshoots

The importance of the study of the regional habit of oreshoots as a basis for discovery or prediction of extension in a given area has been often emphasized. Shoots in a particular district may have characteristic shape; be lenslike, conical, chimneylike, mantalike, etc., or they may bottom at near the same elevation, or they may favor certain beds or intrusives.

For examples I shall again draw on the Parral district. Along the Veta Colorado, and associated veins of the same age, the oreshoots bottom within a few tens of meters of the same elevation. Many of the shoots, furthermore, are V-shaped, with the point of the "V" downward and generally pitching in a common direction.

(III) Study of the Relation of the Accident of Erosion to the Position of Shoots of Lenslike Form

This subject was recently discussed by Blanchard and Boswell.[2] It is especially applicable to lead-silver veins, because their shoots are commonly, if not usually, lenslike. If the form of part of the shoot that has been disclosed by mining be measured, the assumption of lenslike form may aid extension prediction.

(IV) Statistical Studies

Statistical studies of all known, or rather described, oreshoots might make our ideas of extension more quantitative. Thus, shoots of a certain type, such as those of the Tertiary silver veins, bottom at shallow depth. Or do they? What is the average depth, the relation of it to outcrop length, richness, tonnage, or to width of the vein? Hoover[3] has made pertinent suggestions along this line.

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[2] P. F. Boswell and R. Blanchard: Oxidation Products from Sphalerite and Galena. *Econ. Geol.* (1927) **22**, 421.

[3] H. Hoover: The Principles of Mining, 21–33. McGraw-Hill Book Co., New York, 1909.

By such studies it might be practicable to express possibilities of extension in terms of chance or probability. Thus if a shoot is of a certain type, or size, or richness, for example, what is its chance for a given depth extension? Is it 1 in 10, or 1 in 4?

By going through all available literature on veins in Utah, data on the ratio of outcrop length to extension in depth for eight important shoots or lodes were found. The ratios of outcrop length to depth extension were 6:1, 4:1, 3:1, 5:7, 1:2, 1:2, 1:3, 1:4. The average of these is 15:8, which is near the 2:1 ratio suggested by Hoover as the average for a large number of deposits.

(*V*) *Study of Vertical and Horizontal Hypogene, and Vertical Supergene Zoning in Shoots*

The extension and zoning of oreshoots of supergene origin has been rather thoroughly studied in the last 20 years. Hypogene zoning has been somewhat neglected. The suggestion has been made that when shoots change their primary mineral character in depth they are liable to play out shortly. This lead and others should be further investigated.

(*VI*) *Study of the Relation of Oreshoot Extension to the Stage of Erosion of the Source Cupola*

Pioneer work on this subject has been done by B. S. Butler and W. H. Emmons. The theory is that ores are mainly associated with the top of the source cupola and that their chance for extension in depth is dependent on the depth of erosion below the apex of the intrusion. This phase of the subject could well be further studied.

The above are some of the more obvious divisions of the subject of extension. Many others, no doubt, will occur to the reader. For example, many veins split or feather when they are about to play out. In certain districts this is recognized as the end of the ore.

On the Mesabi Range, markers, such as the "algal" horizon, persist through the ore. Where the marker can be followed, it is possible, by measuring the slump, to get a fairly accurate idea of the depth of the ore.

It has been noticed that postmineral faults frequently avoid, and curve around, oreshoots and accompanying walls that have been strongly silicified. It may be useless to drift farther when such a fault is encountered. The fault means the end of the ore, not the offset of it.

Acknowledgments

I am indebted to Augustus Locke of San Francisco, Calif., and A. C. Spencer of the U. S. Geological Survey, Washington, D. C., for criticism of the manuscript of this paper.

Silver-bearing Minerals of Some Ores from the Tintic Mining District

By A. W. Hahn,* Salt Lake City, Utah

(New York Meeting, February, 1929)

The importance, to geologists and metallurgists, of knowing the mineralogical forms in which the silver occurs in ores has increased during the past few years, and will continue to increase as long as working hypotheses are converted into proved theories and rule of thumb methods are supplanted by scientific data.

The methods used for the determination of these silver minerals have been: (1) careful study of polished ore sections with the microscope, and more recently, (2) metallurgical separation of the ground ore by flotation, tabling and panning, supplemented by microscopic examination of these separated minerals, and chemical analyses.

During the course of some experimental work carried on by the testing department of the International Smelting Co. and myself, it became important to determine, in a general way, the distribution of the silver values in the minerals, as they exist in Tintic Standard and North Lilly siliceous oxidized lead-silver ores. These mines are in the East Tintic Mining District, State of Utah.

In this work the second method was used. Differential flotation tests were run, separating the sulfide from the oxidized minerals, and the products were assayed. Various concentrates were carefully panned and the separated products assayed; special samples of ore were selected and assayed. The microscope was used only for the purpose of determining whether the individual minerals were in the so-called "free" state.

ANALYSIS OF TINTIC ORES

These chemical and mineralogical determinations showed the silver to be present as native silver, argento jarosite, argentite, and combined or inherent in the tetrahedrite, chalcopyrite, galena, pyrite, cerussite, anglesite, mimetite, malachite, iron oxide and occluded in the quartz. Careful microscopic work would probably show other forms in small amounts, such as the halide silver minerals and some of the antimony sulfide combinations.

The tabulation of the analyses of the various products (Table 1) shows no evidences of fixed ratios of silver to any particular base metal.

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* Consulting Metallurgist.

TABLE 1.—*Analyses of Products*

Sample Number	Origin of Sample	Assay						Per Cent. of Total in Sample			Remarks
		Ag	Pb	Cu	Fe	As	Insol.	Ag	Pb	Cu	
1	Flotation sulphide concentrate	129.5	33.8	3.20	15.2		10.8	56.7	16.6	70.3	Straight flotation
2	Flotation sulphide concentrate	132.2	22.1	3.87	17.6		15.8	26.4	4.8	35.8	After table concentration
3	Copper sulfides and pyrite	86.2	15.0	3.60	23.4		17.2				Sands panned from flotation concentrate
4	Copper sulfides and pyrite	27.9	19.9	2.40	20.8						Sands panned from flotation concentrate
5	Copper sulfides and pyrite	28.4	10.8	2.90	19.0		19.9				Sands panned from flotation concentrate
6	Slime sulfides and jarosite	102.2	45.0	1.60	6.6		11.6				Sands panned from flotation concentrate
7	Sand sulfides and jarosite	91.9	48.2	1.75	7.2		7.2				Sands panned from flotation concentrate
8	Galena product	28.4	74.2	0.50	5.2						Sands panned from flotation concentrate
9	Tintic Standard pyrite concentrate	3.6	3.5	0.08	30.6		27.1				Show low-silver pyrite
10	Oxidized lead product	30.0	69.5		1.6		1.7				Sands panned from flotation concentrate
11	Oxidized lead product	30.0	65.0		3.0						Sands panned from flotation concentrate
12	Flotation oxide concentrate	23.9	47.4	0.25	5.9		20.4	20.7	46.2	10.9	No. 1 oxide lead flotation concentrate
13	Flotation oxide concentrate	19.0	38.0	0.20	7.5		29.4	15.2	34.4	8.1	No. 2 oxide lead flotation concentrate
14	Flotation oxide concentrate	25.3	38.0	0.76	10.1		26.6	17.3	28.1	24.1	Shows influence of copper
15	Flotation oxide middling	62.5	23.5		24.5		25.6				Shows silver in oxide iron
16	Flotation oxide middling	79.8	5.0	0.14	15.0		58.3				Shows silver in oxide iron
17	Flotation oxide middling No. 16 recleaned	336.9	17.4	0.45	17.0	0.70	37.6				Shows silver in oxide iron and copper
18	Flotation oxide middling	24.7	5.3		5.8		78.2				Shows silver in jarosite form
19	Red oxide ore	18.0	0.8		32.7		48.4				Shows silver in oxide iron
20	Black oxide ore	1.25	27.4		20.3		34.0				Shows low silver in oxide iron
21	North Lilly arsenical	38.7	54.5			0.70					Shows presence of mimetite
22	Tintic Standard sulfide copper concentrate	8385.0	1.6	14.3				71.0	100.0	15.0	Shows unusual high silver in tetrahedrite
23	Tintic Standard oxide copper concentrate	842.0	Nil	18.5				28.0		78.2	Shows lower silver in malachite
24	Siderite	5.2	3.2		22.2		50.0				Shows low silver in siderite
25	Flotation tails	2.34	1.0								Cyanide tails—heads 3.9 silver

The bulk of the silver associated with the sulfide minerals appears to predominate in the copper sulfides, more particularly the tetrahedrite, as shown by comparison of samples 3, 4, 5, and 22. These samples were carefully panned from the sulfide flotation concentrate, except No. 22, which was a specimen sample from the Tintic Standard mine, high in copper and silver, which was further concentrated by flotation. The tetrahedrite predominated in this sample. The bulk of the lead is present as cerussite and anglesite, as shown by samples 11, 12 and 13, which were the oxide concentrates from a flotation operation and contained practically no sulfide minerals, these having been previously removed. The ratios of lead to silver in the galena, and oxidized lead samples where the copper is under 0.50 per cent. varies from 2.6 to 2.0 to 1.

In these samples, where argento jarosite is present, and where the relative percentage of iron oxide is high, the silver assay increases to a remarkable extent. This is shown in an extreme case, samples 16 and 17. Just what this particular red iron mineral is, carrying this silver, was not determined. Selected samples, as No. 19, a red variety, showed rather high silver. The siderite, sample 24, is low in silver, as is a black variety, high in lead, sample 20. Sample 5 shows jarosite silver in the sulfide concentrate. It is probable that silver jarosite is quite widely distributed, and it may be that this form of silver is occluded in the red iron mineral, high in silver, shown in samples 15, 16, 17 and 19.

The pyrite in these samples carries little silver, as evidenced by sample 9, and the varying percentages of iron in the sulfide products show no effect on the silver assay. Sample 9 was a selected sample of pyrite ore from the Tintic Standard mine, concentrated by flotation. The amount of lead in this concentrate could account for the small amount of silver present.

A sample of tailings, assaying 3.9 oz. silver and 1.1 per cent. lead, was leached with cyanide solution. The leached tailings assayed 2.34 oz. silver and 1.0 per cent. lead. The residual silver may have been in the form of jarosite or occluded in the quartz gangue. It has been my experience that nonsulfide forms of silver, insoluble in cyanide solution, are not amenable to concentration by flotation. Jarosite is an exception. The silver insoluble in cyanide is, therefore, a rough indication of its composition.

Distribution of the Silver

It is quite evident that no over-all silver-lead ratio exists in these ores. From the various samples, the indicated ratios are:

	Silver Ounces	Lead Per Cent.	Copper Per Cent.
Galena	1.0	2.6	
Oxidized lead minerals	1.0	2.2	
Sulfide copper minerals	14.0		1.0

Only an intense microscopic study of these products, supplemented by detailed chemical analyses, will determine a reasonably accurate distribution of the silver in these ores. The value of such an investigation as an aid to the successful concentration of these ores by flotation cannot be overestimated. The cooperation of skilled microscopists with those engaged in the metallurgical task of recovering metals from ores by flotation, more particularly silver, is essential, if metal recoveries are to improve. The microscope work should be supplemented by refined chemical determinations of the various products.

A rough calculation, based on analyses, the above ratios and a superficial knowledge of the constituents of these ores, and products from flotation tests, shows a percentage distribution of silver in an ore containing 23.5 oz. silver, 21.0 per cent. lead, 0.5 per cent. copper, 7.4 per cent. iron, 51.0 per cent. insoluble, to be as follows:

	Silver Ounces	Per Cent.
In ore	23.5	100.0
In galena	1.7	7.2
In copper sulfides	5.6	23.8
As argentite and pyrite	3.5	15.0
As jarosite	0.5	2.1
In oxidized lead minerals	8.0	34.0
In iron oxide	3.0	12.8
In quartz	1.2	5.1

In this tabulation, I have taken into consideration the results showing a great increase in silver assay when the copper in the sulfide flotation concentrates showed more than 3.0 per cent. This probably indicates different silver values in the chalcopyrite and tetrahedrite in these products.

CONCLUSIONS

Data developed by this preliminary investigation lead to the following general conclusions:

1. About one-half of the silver is present as a sulfide or associated with sulfide minerals in a relatively small percentage, by weight, of the whole ore. This is true, even in those ores highly siliceous, low in lead and high in silver, showing that argento jarosite is not the predominating mineral, although it is of considerable importance.
2. Tetrahedrite carries relatively large amounts of silver, considering the small percentage of this mineral present in these ores.
3. The lead minerals, both sulfides and oxidized forms, show a low ratio of silver to lead.

4. The silver associated with a dark red siliceous iron oxide is widely disseminated in these ores, and the percentage of this type of silver mineral relatively high.

5. The amount of silver occluded in the quartz gangue, probably does not exceed 2.0 ounces.

ACKNOWLEDGMENT

It is with the cooperation of the International Smelting Co. and the Tintic Standard Mining Co., and through their courtesy, that the results of this preliminary investigation are brought to your attention.

Origin of Iron Ores of Iron Mountain and Pilot Knob, Missouri

By Joseph T. Singewald, Jr.,* and Charles Milton,† Baltimore, Md.

Among the genetically interesting iron ores of the United States are those of the St. Francis Mountains near Ironton and Iron Mountain, Missouri. They are specular hematite in porphyry. The Iron Mountain and Shepherd Mountain deposits are generally spoken of as veins and those of Pilot Knob, Cedar Hill, and Russell Mountain as bedded deposits. Iron Mountain has produced nearly 4,000,000 tons of ore and Pilot Knob about 1,500,000 tons. The output of the other three localities has been insignificant. Pilot Knob is no longer worked as an iron mine, but the rock is now being crushed for use as road metal. Iron Mountain has continued as an active producer. In 1927, the property was acquired by the M. A. Hanna Co. and an extensive prospecting and exploration campaign was undertaken, so that the ultimate production of Iron Mountain will doubtless greatly exceed the figures mentioned.

Previous Theories of Origin

Descriptions of the deposits and theories of their origin have been concerned mainly with Iron Mountain and Pilot Knob, which represent the two contrasting types of occurrence. The Iron Mountain deposits have been considered of close magmatic origin by Whitney[1] in 1854, Geijer[2] in 1915, and Spurr[3] in 1927. Whitney referred to them as of eruptive origin. Geijer considered them fissure fillings at magmatic temperatures. Spurr called them vein dikes formed from ore magma which was essentially fluid iron oxide. Schmidt[4] in 1872 ascribed them to fissure filling by thermal chalybeate waters of meteoric origin. Nason[5] in 1892 advanced essentially the same explanation. One of the best

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[1] J. D. Whitney: The Metallic Wealth of the United States, 433, 479.

[2] P. Geijer: Some Problems in Iron Ore Geology in Sweden and in America. *Econ. Geol.* (1915) **10,** 324.

[3] J. E. Spurr: Iron Ores of Iron Mountain and Pilot Knob. *Eng. & Min. Jnl.,* (1927) **123,** 363.

[4] A. Schmidt: Iron Ores of Missouri. Geol. Survey of Missouri (1873) 93.

[5] F. L. Nason: Iron Ores of Missouri. Geol. Survey of Missouri (1893) **2,** 16.

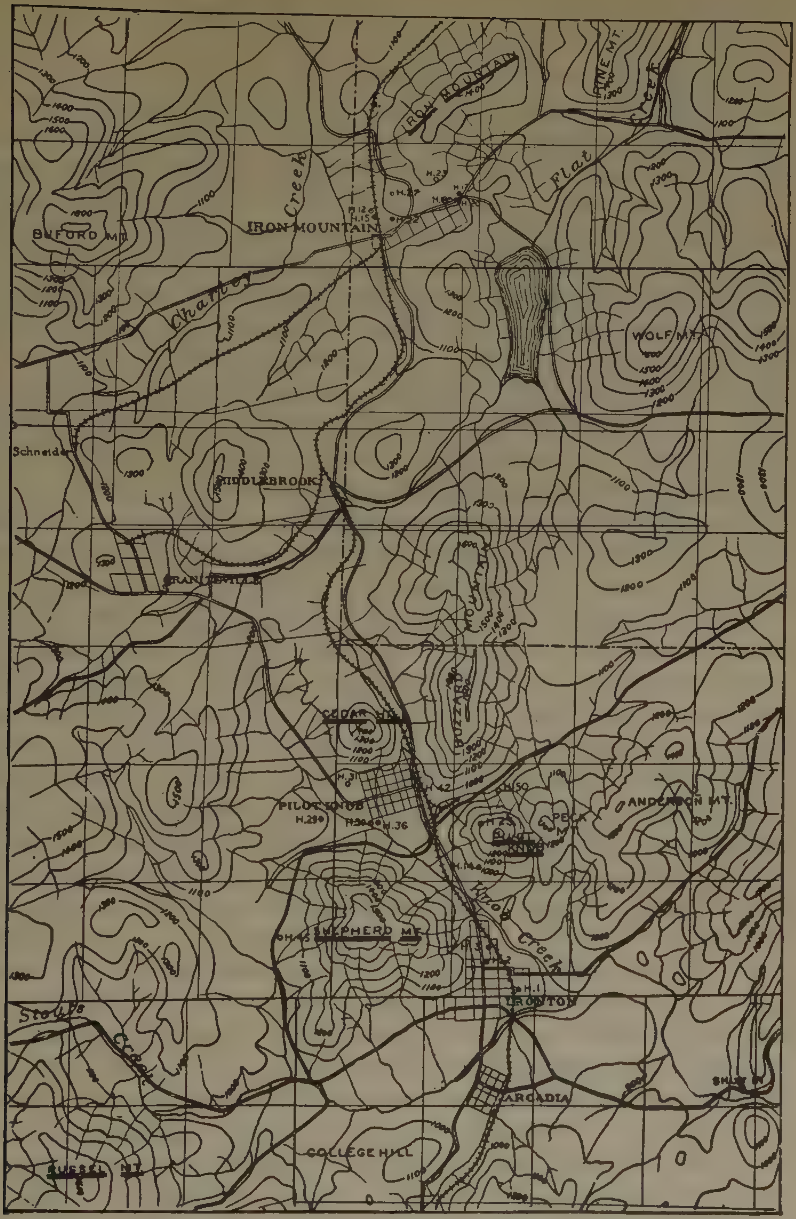

Fig. 1.—Iron Mountain-Pilot Knob district. (After Crane.)

accounts of these deposits is that given by Crane[6] who considers the Iron Mountain deposits fissure fillings and replacements by hot iron-bearing solutions of magmatic origin.

Because of the stratified character of the Pilot Knob ores, their genesis has been considered different from that of the Iron Mountain ores by some geologists. Whitney said that Pilot Knob differs considerably from Iron Mountain in character, but did not specify in what particulars genetically. Schmidt ascribed the Pilot Knob ores to gradual replacement of stratified porphyry by solutions similar to those which he believed deposited the Iron Mountain ores. Nason refers to the Pilot Knob ores as of sedimentary origin, but adds that they were later raised and exposed by erosion and "with the destructive action the more soluble strata were etched away and replaced by iron." Geijer says Pilot Knob forms a striking analogy to Haukivaara. "At both places there is a succession of rocks of probably agglomeratic and tuffogeneous nature, strongly altered through hydrothermal activity, silicified or changed to sericite schists, and impregnated with hematite." Crane considers Pilot Knob a more or less complete replacement and infiltration of stratified tufaceous beds by iron oxide in solutions of magmatic origin. Spurr considers Pilot Knob a black sand beach deposit.

In his review of previous theories of origin, Spurr says he is in accord with Whitney and Geijer and at variance with Crane concerning Iron Mountain and aligns himself with Nason concerning Pilot Knob. Geijer, however, says he differs from Crane only as to the relative importance of fissure filling and replacement in the localization of the Iron Mountain ores, and that "this unimportant detail excepted, Mr. Crane's results are absolutely the same as I have reached concerning the nearest Swedish relatives of these deposits." Spurr and Whitney stand alone, therefore, in emphasizing the idea of an injected ore magma in the case of Iron Mountain. At Pilot Knob, Spurr emphasizes the detrital sedimentary character of the ore to a greater extent than Nason. Spurr makes a greater genetic difference between Iron Mountain and Pilot Knob than any of the other geologists. Crane considers both genetically similar and differing mainly in the nature of the loci of mineralization. Geijer is essentially in agreement with Crane.

In the summer of 1927, the senior author had the privilege of spending a few days in southeastern Missouri and the opportunity of a brief visit to Iron Mountain and Pilot Knob. His field observations inclined him to the views of Crane and to disagree with Spurr, particularly regarding the sedimentary character of Pilot Knob. A microscopic study of the specimens he collected and of a set of slides and hand specimens kindly lent for study by Prof. W. S. Bayley, which included also material from Shepherd Mountain and Cedar Hill, confirmed these conclusions.

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[6] G. W. Crane: The Iron Ores of Missouri. Missouri Bur. Geol. and Mines [2] (1912) **10,** 107.

IRON MOUNTAIN

It is not intended in this paper to give a geologic description of these Missouri iron-ore deposits, as they are described in the writings of the geologists already cited. Only such data will be presented as are of significance in arriving at a conclusion concerning their origin. All authors have been in agreement in emphasizing the veinlike character of the Iron Mountain deposits. Crane believed that they are in part fissure filling, but more largely a replacement of the porphyry. This study of the relations of the ore to wall rock and to gangue also leads to an emphasis upon the replacement character of these orebodies and to subordinate their character as cavity fillings.

The Iron Mountain porphyry ranges from a felsite to a felsite porphyry with pronounced flow bedding and some pyroclastic structure. The rock around the walls of the old open cuts has its feldspar phenocrysts changed largely to fine-grained aggregates of quartz and chlorite. Streaks and veinlets of ore traverse the rock in such an intricate manner and with such wavy boundaries as to be explained only as having made their way largely by replacement. A thin section of pyroclastic rock with fragments up to 10 mm. dia. has a matrix of hematite and some of the fragments are considerably eaten into by hematite. The slides afford excellent examples of all degrees of replacement. Some of the veinlets consist of well crystallized hematite with the interstices filled with chlorite, calcite, and garnet. In the mine larger masses of olive-green garnet rock were seen. This rock consists of garnet, hematite, quartz, calcite, and an amphibole. The hematite tends to be later than the garnet and to replace it along their contacts. It is also later than and replaces the calcite. The calcite has an unusual spherulitic development. Cutting the garnet are small veinlets of calcite and quartz. The quartz occurs as a fine-grained mosaic. The amphibole occurs as swarms of minute pale green prisms in the quartz. In one section a peculiar apatite was seen analogous to the carbon apatite described by Brauns.[7] This consists of a normal center with a peripheral zone of three to six individuals, each showing zoning and parting. Normal apatite also occurs. The sections of Iron Mountain ore differ from those from Pilot Knob, Shepherd Mountain, and Cedar Mountain in a higher ratio of hematite to quartz in the introduced minerals.

As constituents of the Iron Mountain porphyry were recognized orthoclase, plagioclase, quartz, hematite, sphene, chlorite, epidote, and garnet. Directly associated with the introduced hematite are quartz, apatite, garnet, tremolite, and calcite. This assemblage is indicative of high-temperature mineralization.

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[7] E. Brauns: Über den Apatit aus der Laacher See Gebiet. *Neues Jahrb. f. Min.* (1917) **41**, 72.

SHEPHERD MOUNTAIN

According to the published descriptions of Shepherd Mountain, these deposits are more pronouncedly fissure fillings. The country rock ranges from a felsitic to porphyritic rhyolite and exhibits marked flow structures. The original rock consisted mainly of quartz and feldspar with small amounts of accessory minerals. Most of the feldspar was orthoclase, but locally acid plagioclase was abundant. Much of the micaceous material is probably derived from feldspar. Accessory minerals in the rock are zircon, green mica, chlorite, rutile(?), apatite, sphene, fluorite, barite, epidote, and calcite. One slide shows veinlets of calcite, epidote, and chlorite in which chlorite seems to be the youngest mineral.

Replacing the porphyry are veinlets of hematite and chlorite. In one specimen, large quartz grains are traversed by small sinuous veins of hematite and chlorite. The larger veins in the hand specimens consist mainly of hematite, chlorite, and quartz. The periods of formation of the three minerals overlapped considerably, but the chlorite and hematite were introduced more or less contemporaneously and the quartz is in part younger. The striking feature of the Shepherd Mountain collection is the greater abundance of quartz than at Iron Mountain and the association of hematite and chlorite.

The mineral assemblage and the relations of the ore veinlets to the country rock suggest hypothermal mineralization rather than magma injection at Shepherd Mountain.

CEDAR HILL

The rock associated with the Cedar Hill ore is pyroclastic and ranges from tuff to agglomerate in texture. The fragments consist of felsite and quartz porphyry and are embedded in matrix of more finely comminuted material of the same character. The principal minerals of the rock are quartz, orthoclase and muscovite.

Hematite has extensively replaced the matrix of the fragmental rock, and accompanying the hematite is more or less quartz. These minerals have also penetrated and replaced some of the larger rock fragments. Numerous replacement veinlets of hematite traverse the groundmass where it has not been so completely replaced. Many of the veinlets contain, besides hematite, quartz, mica and chlorite. Other veinlets cutting the hematite matrix are of quartz or orthoclase, or both. Veinlets of orthoclase grade over into quartz-orthoclase veinlets. One vein has a center of bowed plates of hematite, and fractures and cracks in the orthoclase and quartz are filled with hematite.

The Cedar Hill suite of specimens shows the replacement of the matrix of a tuff and agglomerate and to a limited extent the larger fragments by hematite. The latter in turn is cut by younger quartz and orthoclase veinlets with or without hematite. The phenomena indicate high-temperature mineralization under pegmatitic conditions.

PILOT KNOB

No more concise word picture of the Pilot Knob occurrence can be drawn than the description by Haworth.[8] He says: "Imagine a steep conical hill rising more than 650 ft. above the valley . . . Imagine further a plane inclined about 13° SSW. passing through the hill, about 150 ft. below the summit, and you have the approximate location of the ore bed." The section of the hill lying above this plane as given by Crane is:

	FEET
Ferruginous porphyry breccia (formerly called conglomerate)	100
Upper ore bed	10 to 20
Lower ore bed	6 to 30
Footwall porphyry	

The lower bed is described by Spurr as solid shaly hematite, showing ripple marks, mud cracks, and rain prints. He concludes that it is a beach deposit. The upper bed, he says, is finely stratified poorer iron ore containing porphyry fragments. The overlying ferruginous porphyry breccia is described by him as rudely stratified poor iron ore, with angular, subangular, and rounded fragments of porphyry and some hematite in a matrix largely of iron.

One problem involved in the interpretation of Pilot Knob is the determination of the character of the more or less stratified materials forming the orebody and the overlying leaner rocks. Are they sedimentary clastic materials or are they pyroclastic tuffs and agglomerates? Spurr assumes the former. If that is correct, the explanation of the origin of the ore still involves the additional problem of whether the ore is syngenetic or epigenetic. Crane assumes that the rocks represent tuffs and agglomerates into which the ore has been subsequently introduced.

If Crane is correct in his interpretation of the character of the rocks, the ores can not be of sedimentary origin. The decision as to the syngenetic or epigenetic character of the ore rests very largely on that concerning the origin of the rocks. That is, the mineralization occurred either at the time of that at Iron Mountain, as a later magmatic phase of the igneous activity represented by the porphyries and their associated pyroclastics, or it represents the sedimentary accumulation of hematite derived from that deposited during the aforementioned mineralization.

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[8] E. Haworth: The Age and Origin of the Crystalline Rocks of Missouri. Geol. Survey of Missouri *Bull.* 5 (1891) 17.

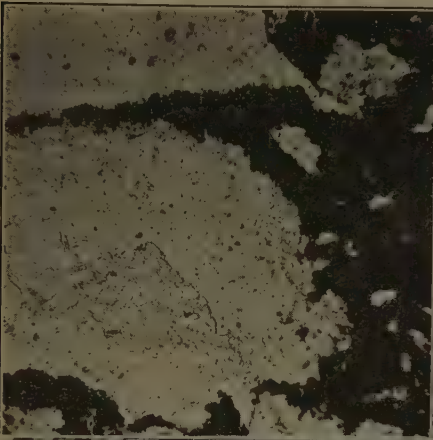
It is not likely that there occurred a later period of epigenetic high-temperature mineralization so analogous to that of Iron Mountain.

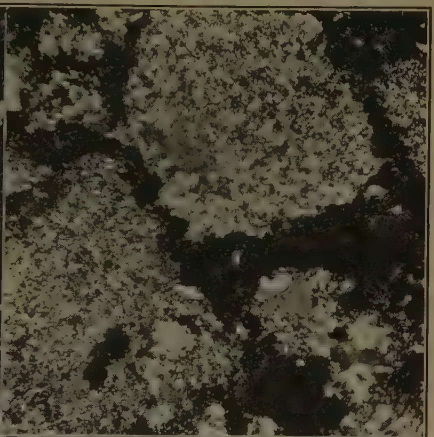
In view of his application of the vein-dike theory to the Iron Mountain deposits, Spurr was forced to place an entirely different interpretation upon Pilot Knob, as it is obviously not a vein dike. Though considering Pilot Knob a "black-sand" beach deposit, he adds, "Only in Japan, I believe, have they been commercially worked as iron ores." Yet Pilot Knob has produced 1,500,000 tons of ore! The probability is, therefore, that it is not a black-sand deposit. No certain evidence is advanced to establish the sedimentary origin of the rocks as against stratified tuffs and agglomerates. The highly unsorted nature and angular character of the constituents of the coarser rocks is more suggestive of pyroclastic material. The relations of ore to gangue and the constituent minerals demonstrate epigenetic, high-temperature mineralization.

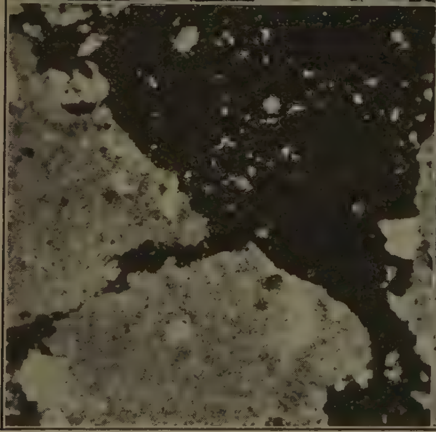
The textures of the Pilot Knob rocks range from that of the fine-grained, finely banded ore to that of the coarse overlying fragmental rock. The fine-grained rock consists mainly of hematite with greater or smaller amounts of quartz and small flakes and tufts of muscovite. One of the specimens that shows most pronounced and regular bedding consists of a mosaic of quartz, muscovite, and hematite with an abundance of minute needles of blue tourmaline. The interlocking quartz grains and the shape of the hematite grains are not at all suggestive of a quartz and hematite sand. In the leaner ore with larger rock inclusions, hematite not only replaces the fine-grained matrix, but has been deposited at the expense of the larger inclusions. All stages of replacement of the inclusions are seen. These phenomena extend out to the lean overlying porphyry agglomerate in which many of the larger fragments present irregular, ragged boundaries against the hematite of the matrix, affording unmistakable evidence of replacement of rock by ore. Further evidence of replacement is afforded by phenocrysts of quartz that are corroded by hematite. Some of this quartz has been magmatically corroded by the felsitic groundmass and this in turn has been replaced by hematite, leaving the corroded quartz phenocrysts now embedded in ore.

These replacement phenomena are illustrated in the photomicrographs of Figs. 2 to 5. The relations are not those of a detrital deposit made up of water-borne grains of hematite and felsite sand.

Of especial significance is the occurrence of blue tourmaline. It occurs sparingly but quite widely distributed in both the ore and in the hanging-wall rock. Other minerals that were noted in small amounts are zoisite, sphene, barite, and a very little apatite. The hematite matrix is cut by replacement veinlets of quartz, barite, hematite, and mica. These veinlets are traversed in turn by still later barren quartz veinlets. There are also veinlets and small areas of pinitic material presumably derived from hydrothermal alteration of feldspathic material. Locally it occurs in larger dense porcellanic masses.







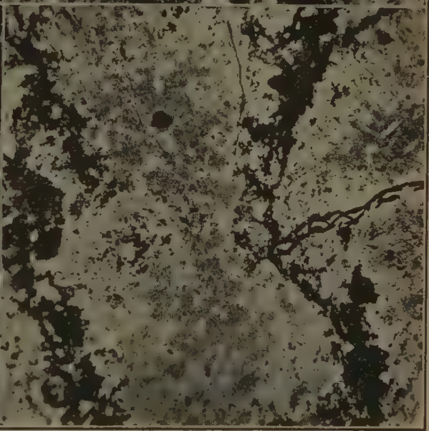


Fig. 2.—Pilot Knob ore. ×120.
Hematite replacing quartz-sericite aggregates. The tongues of hematite penetrating the rock are bordered by very narrow margins of sericite.

Fig. 3.—Pilot Knob. Hanging-wall rock. ×35.
Crossed nicols. Hematite replacing matrix and corroding felsitic fragments of pyroclastic rock. The rock fragments are largely a quartz-sericite aggregate with larger quartz phenocrysts.

Fig. 4.—Pilot Knob. Contact between tuff and rhyolite. ×40.
Hematite replacing fine-grained felsitic rock. Quartz phenocrysts are more resistant to replacement than the felsitic groundmass.

Fig. 5.—Pilot Knob. Hematite in felsite. ×40.
Ragged veinlets of hematite traversing fine-grained felsitic rock showing irregular and ragged replacement boundaries. The gray and mottled appearance of the rock is due to inclusion of swarms of flakes and dust of specularite.

The occurrence of hematite is not limited to the ore beds and hanging-wall rock; it has also been deposited some distance into the footwall porphyry. A thin section of rock 25 ft. below the footwall shows a pyroclastic rock with hematite bunched around the fragments and in irregular tongues projecting into the fragments, suggesting subsequent infiltration and replacement of the matrix and fragments.

The microscopic evidence proves that the hematite in its present occurrence was deposited from hot solutions capable of depositing tourmaline, which penetrated not only the ore bed but the overlying fragmental rock and to less extent the underlying more compact porphyry. If it existed originally as black-sand grains, these solutions dissolved it, and reprecipitated it at the expense of the gangue with which it was associated, probably carrying some of it upward to be deposited in the hanging-wall rock and carrying some down into the footwall rock. If so much solution and redeposition took place, it seems surprising that the sedimentary banding was preserved so perfectly. If, on the other hand, the hot solutions brought in the iron, and deposited it by replacement of the rock, one can understand why the amount of mineralization is proportional to the degree of comminution of the rock particles, and why maximum mineralization took place in the fine-grained tuffs, less in the coarse agglomerates, and still less in the denser footwall porphyry. The well-preserved banding of the ore is then the preservation of banding in the tuff. The localization of the ore in the tuff is due to the deposition of more hematite in the more easily replaced tuff layers.

Origin and Nature of Mineralization

In the preceding pages, evidence has been set forth to prove that all of these Missouri iron-ore deposits were formed at high temperature, at the temperature of consolidation of pegmatite or thereabouts. All are characterized by more or less replacement of the country rock. In the Iron Mountain type fissure filling played an important role. In the Pilot Knob type the mineralization followed porous, stratified beds of pyroclastic rock and replacement of constituents of those rocks was the determining cause of ore localization. The associated minerals are those more commonly formed by the action of solutions and mineralizers on rocks than as a result of reaction between invading magma and rock, indicating that the ores were introduced under high-temperature hydrothermal or very mobile pegmatitic conditions rather than as ore magma injections. These facts indicate beyond peradventure a magmatic source for the mineralizers. Their predominant constituents were those of hematite and quartz. They differed in the different deposits mainly in the ratio of their ferric iron and silica content, and the ratio of these constituents also varied in the same deposit during the period of mineralization.

The country rock porphyries of the iron ores are part of an extensive granite-porphyry area in southeastern Missouri. The relationship of the granite and porphyry have been given considerable attention in the literature on the geology of Missouri, but a further detailed study is required to be able to arrive at a conclusive decision. The porphyry has been regarded as an upper peripheral facies of the granite and the one rock considered to grade into the other. Not enough was seen of the contacts between the rocks to venture to settle this question. However, E. T. Campbell, geologist of the St. Louis Smelting & Refining Co., who has displayed much enterprise in extending the boundaries of his geologic interest in the region beyond the limits of the rocks containing the lead ores, showed some outcrops on Knob Lick Mountain near Syenite, about 12 miles east of Iron Mountain, where the granite is intrusive in the porphyry. Several small granite quarries on this mountain show a perfectly sharp contact between granite and porphyry. The granite contains porphyry xenoliths and dikelets of granite cut the porphyry. A close relationship between the granite and the porphyry is suggested by the common feature of an unusual abundance of minute specular hematite inclusions in the minerals of both rocks, a phenomenon that suggests derivation of them and of the iron ores from a common magma. It appears probable, therefore, that the iron ores were introduced by iron-silica-rich end products of a deep-seated granitic magma from which the porphyry and granite had been previously derived.

Summary

The subject of the origin of the specular hematite deposits of southeastern Missouri was recently reopened through the expression by Spurr of views at variance with those of Crane which had been endorsed in the main by the observations of Geijer. This contribution to the subject presents evidence in support of Crane's theory. The microscopic study of the ore and associated rocks demonstrates a close genetic relationship between Iron Mountain and Pilot Knob. The fuller description of the mineralogy of the ores presented in this paper, notably the occurrence of tourmaline at Pilot Knob, proves that all of the deposits were formed at high temperature. It is suggested that porphyry, granite, and ore deposits represent successive derivations from a parent granitic magma. The peripheral relations of the Iron Mountain deposits and the replacement nature of Pilot Knob preclude the application of the vein-dike idea to these deposits. Ore deposition has been so largely by replacement that the activity of highly mineralized magmatic derivatives must be accepted. If Iron Mountain alone were involved, one might be inclined to consider the mineralization pneumotectic, but since the Pilot Knob mineralization is so nearly like it, a more remote magmatic affiliation is indicated; a mineralization for which the term hypothermal would seem more correct.

DISCUSSION

E. T. CAMPBELL, St. Louis, Mo. (written discussion).—My observations, from many trips to Pilot Knob, as to the origin of the ores are in accord with those of Crane and hence with the authors of this paper. They offer the best explanation for the various grades of ore, taking into consideration the nature of the country rock in which it is found. I know of one other locality about six miles south of Pilot Knob where well-stratified water-worked volcanic tuff is exposed in a road cut. This material is not mineralized but is quite comparable to what is presumed to have been the nature of the main 20-ft. orebody on Pilot Knob before it was impregnated with the hot iron-bearing solutions.

Relation of Electrode Potentials of Some Elements to Formation of Hypogene Mineral Deposits*

By B. S. Butler,† Tucson, Ariz., and W. S. Burbank,‡ Washington, D. C.

(New York Meeting, February, 1929)

Study of the ore deposits of Colorado has disclosed, in numerous places, sharp changes in both mineralogy and metal content of the primary or hypogene deposits with change in depth. A clear understanding of the meaning of these changes is requisite to form intelligent conclusions that can be applied in the exploration of such ore deposits. This paper is an attempt to set forth some of the geologic and chemical relations of minerals that bear on this problem.

The effect of the differing electrode potentials of elements on the order of oxidation and on the order of displacement of one metal by another metal in supergene sulfide deposits has been studied for many years by several investigators. The results all agree rather closely with the original Schürmann series of replaceability of metal sulfides, as Wells has shown.[1]

So far as the writers are aware there has been little attempt from the geologist's viewpoint to correlate the electrode potentials of the elements, other than those of the common metals, with the positions and mode of combination of the elements as found in ore deposits, and in particular in hypogene deposits. Wells[2] discusses electrochemical activity between solutions and certain ore minerals and mentions the possibility of its effect in precipitation reactions.

The senior writer for a number of years has emphasized the importance of oxidation-reduction reactions as a factor in the formation of certain hypogene ore minerals, and as the electrode potentials are a measure of the relative oxidizing and reducing power of the elements and their ions, it seems possible that a study of them may offer some suggestions on certain problems. The data at present available for such a line of attack on oxidation-reduction reactions are too far from complete to offer much help, nevertheless the table of standard electrode potentials of the uncombined elements brings out certain relations of elements to their functions in hypogene deposition that appear not to have been fully appreciated. The purpose of this paper is to indicate certain of these relations, with the hope of stimulating discussion by geologists and chemists. No attempt is made to explain all the relations observed; the aim of the writers is simply to present a viewpoint which may prove of interest to others.

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* Published by permission of the Director of the United States Geological Survey and the Chairman of the Colorado Metal Mining Association.

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[1] R. C. Wells: Electric Activity in Ore Deposits: U. S. Geol. Survey *Bull.* 548 (1914).

[2] R. C. Wells: *Op. cit.*, 30.

Fractional Precipitation of Some Ore-forming Compounds at Moderate Temperatures. U. S. Geol. Survey *Bull.* 609 (1915) 22.

Oxidation-reduction Reactions

It is pointed out in a number of treatises[3] and textbooks on physical chemistry that the electrode potentials are fundamental properties of the elements and provide a measure of the "tendency of electron shells to give off electrons." By plotting the known potentials of the elements, or their ions, against their atomic numbers we obtain an arrangement such as is shown in Fig. 1, in which the periodicity of this property is clearly brought out with its relation to the more recent theories of atomic structure and periodic groups. In each of the four atomic series beginning with the rare gases helium, neon, argon, and krypton, the elements for which accurate experimental data are known fall in the order of decreasing electrode potential, except that in the two series of 18 elements each beginning with argon and krypton sharp jumps in the electrode potentials occur at zinc and cadmium. This sudden change can be correlated, however, with the theories of atomic structure which postulate that the zinc and cadmium atoms are at the head of a group of elements that contain a more stable arrangement of the atomic kernel than the preceding groups of elements, such as that from titanium to cupric copper, which are of variable valence.[4] On the basis of the theories of atomic structure and the position of the elements in atomic groups, the writers have taken the liberty of interpolating the positions of some elements for which experimental data on electrode potential are lacking. Such an estimation, is quantitatively rough, of course, but it should give us an idea of the relative positions of some elements.

An inspection of Fig. 1 brings out several relations with regard to the stability of compounds of the elements in hypogene ores. Let us keep in mind that the elements of high electrode potential are in general the strongest reducing agents, and conversely those of lowest potential are the oxidizing agents, and consider the distribution and compounds of the elements as found in hypogene (primary) ore deposits. The elements with high potentials, +3 to +1, form oxygen compounds—as silicates, carbonates, and sulfates—and with few exceptions are not known as sulfides. The potential of vanadium, an apparent exception, in the valence in which it combines as sulfide may fall below +1. The silicates in ores form most abundantly early, at the high temperatures, but continue to form in decreasing amounts at relatively low temperatures. The carbonates form most abundantly later than the maximum for the silicates, and the sulfates probably have their maximum somewhat later than the carbonates. Both carbonates and sulfates, as well as the silicates of some elements, continue to form to the later stages of ore formation.

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[3] H. S. Taylor: A Treatise on Physical Chemistry, **2**, 803. New York, 1925. D. Van Nostrand.

[4] G. L. Lewis: Valence and the Structure of Atoms and Molecules, 63. New York, 1923. Chemical Catalog Co.

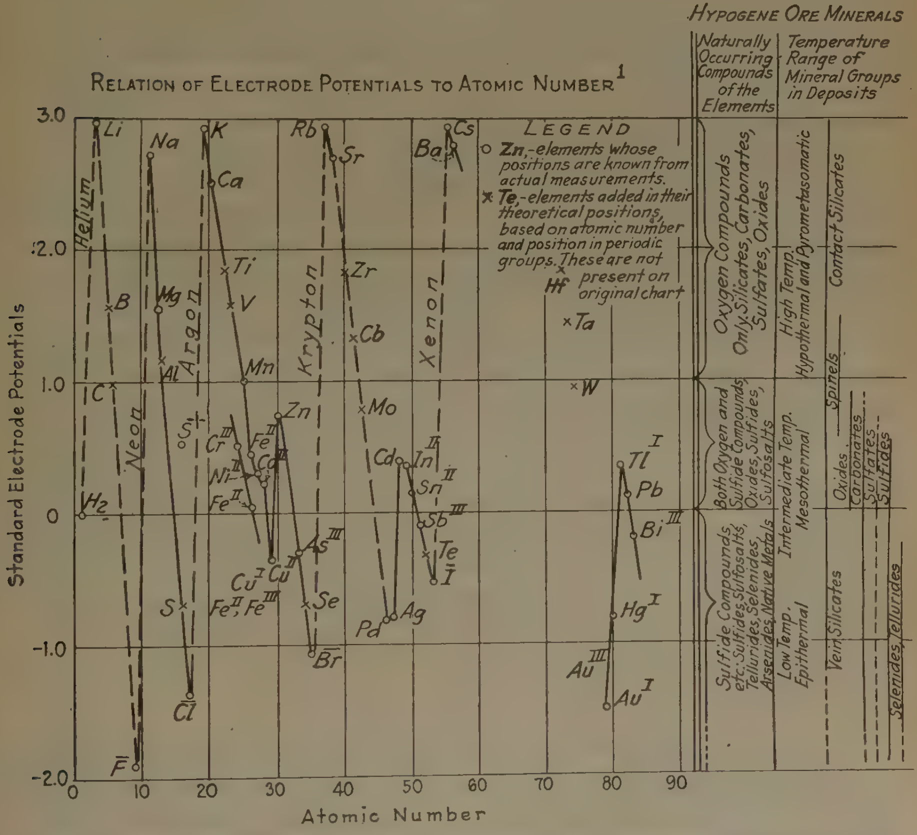

FIG. 1.—KNOWN POTENTIALS OF ELEMENTS PLOTTED AGAINST THEIR ATOMIC NUMBERS. (Chart from treatise by Taylor—*Op. cit.*—with additions by the authors.)

The metallic elements with negative potentials do not form oxygen compounds in hypogene deposits but only sulfides and allied minerals, as tellurides and selenides, or native metals. The metals which occur native most commonly are of low potential.

The elements of the intermediate group, those with potential between zero and +1, form both oxygen compounds and sulfides. Their compounds are formed over a wide range of temperature. At the highest temperatures they form silicates, and with decreasing temperature some form first the spinels, as magnetite, franklinite, and gahnite; then the oxides, hematite, cassiterite, and zinkite; and finally the sulfides. Iron, for example, forms silicates, spinels, oxides, and sulfides; zinc forms similar compounds, which also have a wide range of temperature of formation.

The range of temperature of deposition of a metal is the wider if it occurs in combination with another metal: iron sulfide, for example, begins to form relatively early, and copper sulfide (chalcocite) forms later, but the sulfides of iron and copper together, such as chalmersite, chalcopyrite, and bornite, form at intermediate stages, and the range of both iron and copper is thus extended. Likewise the addition of iron to zinc sulfide seems to change its position of formation toward the higher temperature range characteristic of iron sulfide. Similar relations might be pointed out for other elements.

In contrast to these we may consider lead, which forms very largely as the simple lead sulfide, galena. The metal lead seemingly has a narrower range than iron, copper, or zinc, but the sulfide, galena, is probably not more restricted than chalcocite or pure sphalerite. The combination of iron, copper, zinc, and lead with some of the less common elements, such as bismuth, antimony, arsenic, tellurium and selenium, further extends their range of deposition.

Three Main Divisions of Elements

It appears possible from this distribution in hypogene deposits to divide the elements in the periodic or potential groups into three main divisions—one including all those whose potentials lie above manganese and which form oxygen or more complex silicate compounds almost exclusively; a second including those whose potentials lie between zero and +1 and which form both oxygen and sulfide compounds; and a third consisting of electronegative elements, which form mainly sulfide compounds, sulfosalts, native metals, tellurides, etc.

In order to establish more definitely the positions where such arbitrary divisions might be placed, the elements were arranged roughly in the order of their known or theoretical positions in the potential series, together with the common groups of minerals formed in hypogene deposits. This arrangement is shown in Table 1, in which the mineral groups are placed approximately in the order in which they begin to form, ranging from high to low temperatures. Mineral groups having a wide range of temperature and containing certain "persistent" minerals are placed in positions corresponding to their higher temperature of formation. The position thus assigned to the native elements may be somewhat out of harmony with those of the other groups, but for lack of a

TABLE 1.—*Elements Arranged in Order of Their Known or Theoretical Positions in the Potential Series*

NATURALLY OCCURRING COMPOUNDS OF THE ELEMENTS

HIGH TEMPERATURE IN HYPOGENE DEPOSITS LOW TEMPERATURE

	Alumino-silicates	Zircono-silicates	Titano-silicates	Titanates	Borosilicates Borates	Columbates Tantalates	Fluorides	Simple silicates	Spinels	Oxides	Carbonates	Sulfates	Sulfides	Arsenides	Antimonides	Bismuthides	Sulfosalts of arsenic	Sulfosalts of antimony	Sulfo salts of bismuth	Tellurides	Selenides	Native metals	
Li	X																						OXYGEN COMPOUNDS ONLY
K	X																						
Rb	x																						
Cs	x																						
Na	X	x	x		x	x	x																
Sr	x										x	x											
Ba	x		x		x						x	x											
Ra(?)																							
Ca	X	x	X	x	X	x	X	X			x	x											
Y(?)	x	x	x		x	x	x	x															
La(?)	x	x			x		x	x			x												
Be(?)	x							x															
Sc(?)																							
Ce(?)	x	x	x	x	x	x	x	x															
Th(?)						x		x															
Zr(?)	x	x	x		x	x		X		x													
Ti(?)	x	x	X	X		x				X													
U(?)						x		x		x													
Mg	X			x	x		x	X	X	x	x												
V^{IV}(?)	x							x															
Cb(?)					x	x																	
Ta(?)			x	x	x	x																	
Cr^{II}(?)	x							x	x														
Al	X	x	X		x		x	X	X	X													
Mn	x		x	x	x	x		X	X	x	x		x										OXYGEN AND SULFIDE COMPOUNDS
Sn^{IV}(?)					x	x		x		X													
V^{III}(?)																							
Mo(?)													X										
W(?)						x							x										
C(?)										X			/									x	
Si(?)										X													
Zn					x			x	x				X										
Cr^{III}	x							x	X														
S^{--}																							
V^{II}(?)													x										
Fe^{II}	X	x	x	x	x	X		X	X		X		X	x			x	x				x	
N																							
Ga													x										
Cd										x			X										
In													x										
Tl^{I}													x				x	x		x	x		
Co^{II}								x		x			x	x			x	x	x				
Ni^{II}								x		x			X	X	x		x	x	x	x	x	x	
Sn^{II}													x					x	x				
Pb								x					X				X	X	x	x	x	x	
Fe^{III}	x				x			x	X	X			x										
Ge(?)													x						x				
H										X	x	x	X									x	
P(?)																							SULFIDE COMPOUNDS, NATIVE METALS, ETC.
Ru(?)													x									x	
Sb								x					X					X				x	
Bi^{III}													x						X	x	x	x	
As^{III}													X				X					x	
Cu^{II}								x					X				X	x	x	x	x	x	
Te(?)																							
Cu^{I}								x					X	X	x		X	x	x	x	x	X	
Rh(?)																						X	
I																							
S(?)																						x	
Se(?)																						X	
Hg													X					x		x	x	x	
Ag													X		x	x	X	X	x	x	x	X	
Pd(?)																						x	
Au^{III}																							
Au^{I}																				X	X	X	
Os(?)																						x	
Ir(?)																						x	
Pt(?)														x								X	

LEGEND X Abundant or common minerals of the elements
x Less common minerals of the elements

position better established they are placed last. Although there may be some difference of opinion as to the relative positions of certain mineral groups, and particularly as to the potential arrangement of the elements, they are placed in the positions which available data and the judgment of the writers indicate to be the most probable.

The known occurrences of elements in mineral groups, together with estimates of their importance, are taken mainly from tables by Clarke and Washington,[5] with certain additions and changes tending to make the table applicable solely to primary occurrences. The minerals are arranged in a band extending diagonally across the table. This gives what is essentially the stability range of minerals in ore deposits. The elements that fall between hydrogen and manganese are those of the transition group which form both oxygen compounds and sulfide compounds. These divisions correspond closely to those made by Clarke and Washington[6]—petrogenic elements, metallogenic elements, and transition elements—except that in the divisions adopted here the heavy metals, such as iridium and platinum, fall into the metallogenic group, and the transition group is much larger. However, the limits of the transition group as indicated in Table 1 were selected mainly from the occurrence of the metals with a view to assist in bringing out the stability relations in mineral deposits rather than to exhibit the occurrence of the elements in the earth's crust as a whole.

POTENTIAL OF SULFIDE ION

If it is true, as indicated by the preceding discussion, that the metals of the transition group occur in oxidized condition or as oxygen compounds in the vein-forming stage preceding the precipitation of metal sulfides, does it follow that the sulfide ion (S^{--}), with its high oxidation potential, which falls in this group in the potential series, could have been stable under the same conditions in this extremely low state of oxidation?

Let us consider two specific oxidation reactions:[7]

$$S^{--} = S^0 \text{ (rhombic)} + 2E^- \qquad E^0 = 0.51 \qquad [1]$$

$$Fe^{++} = Fe^{+++} + 1E^- \qquad E^0 = -0.747 \qquad [2]$$

where E^- represents electrons exchanged during the reaction, and E^0 the standard electromotive force of the reaction.

Under conditions comparable to those of the experimental data on potentials, and under oxidizing influences that were favorable to either one, reaction 1 would take place before reaction 2—that is, the sulfide ion would be oxidized to free sulfur before the ferrous ion would be oxidized to the ferric ion. This is evident from the fact that the oxidation or electrode potentials are a relative measure of the "free energy" changes occurring in reactions, and those reactions with the greatest driving force or potential involve the greatest loss in "free energy." It is recognized, of course, that electrode potentials vary with temperature, pressure, concentration, and nature of the solution. But relative potential can hardly be affected seriously by pressure except in reactions involving gases, nor should ordinary changes in concentration and nature of the solutions alter the order of potentials, even though certain potentials may change their positions more rapidly than others. In short, positions of the elements in atomic groups appear so closely related to their potentials, as shown by the chart of atomic numbers, that it is hardly to be expected that ordinary geologic processes would greatly alter the relative positions of the elements in any given environment. On the other hand, as so little is actually known about the nature of ore-forming solutions, predictions of specific reactions in these solutions can as yet hardly be made. The antipathy between ferric compounds and sulfide compounds has been emphasized by several writers,[8] and it seems possible from this antipathy that a relation between the oxidation potentials of the sulfide ion and the ferrous ion may also exist under the conditions prevailing in ore-forming solutions. As no accurate data for the oxidation potentials of sulfur (S^0) to its oxides could be found for conditions comparable to those of the standard electrode potentials, these can not be compared with other oxidation potentials in the series. It is probable, however, that these potentials fall low in the series and that only under relatively strong oxidizing conditions would any appreciable quantities of SO_3 or sulfuric acid be formed. It is well known that certain sulfates, such as barite, are associated with sulfides in some primary ore deposits, and in many such deposits the deposition of barite preceded that of the sulfides. Clearly, then, some of the sulfur was in a very highly oxidized condition prior to the deposition of sulfides.

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[5] F. W. Clarke and H. S. Washington: The Composition of the Earth's Crust: U. S. Geol. Survey *Prof. Paper* 127 (1924) 94, 95.

[6] F. W. Clarke and H. S. Washington: *Op. cit.*, 72.

[7] G. L. Lewis and M. Randall: Thermodynamics, 411. New York, 1923. McGraw-Hill Book Co.

If sulfur occurs in its more oxidized states—that is, as S, SO_2, or SO_3—at the higher temperatures in vein solutions, as suggested above, it is obvious that metals could not be transported mainly as complex sulfide ions in alkaline sulfide solutions. More highly oxidized solutions, such as sulfite, sulfate, or alkali carbonate or bicarbonate solutions, would seem to be more likely carriers. Is it not then reasonable to infer that, at least in certain vein-forming "solutions," perhaps, especially those of intermediate depth, or more shallow types in which the acidic constituents may attain greater concentration, most of the metallic elements at high temperature were transported in a relatively oxidized combination and that they were precipitated as sulfides only with a decrease in strength of the oxidizing conditions for sulfur and metals?

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[8] B. S. Butler: Primary (hypogene) Sulfate Minerals in Ore Deposits. *Econ. Geol.* (1919) **14**, 601, 602.

Some Relations between Oxygen Minerals and Sulphur Minerals in Ore Deposits: *Idem* (1927) **22**, 233.

G. Gilbert: The Significance of Hematite in Certain Ore Deposits. *Econ. Geol.* (1926) **21**, 560.

Basis for Further Investigation

This argument is concerned with the compounds of iron and sulfur found in mineral deposits, because these elements, at the present stage of our knowledge, most readily lend themselves to the purpose. With increase in knowledge of the potentials of other reactions occurring in ore solutions and pertaining to other elements than iron and sulfur, it will be possible to arrive at a better understanding of the stability relations of minerals in hypogene ore deposits. It is with this hope that the writers have ventured somewhat out of their fields and attempted to present and review the data now available, so that they may come to the attention of physical chemists and those better prepared to apply physicochemical laws and experimental data to the solution of such problems. It is recognized that solubility and other factors enter into the problem, but in this paper the effects of electrode potential are emphasized.

The foregoing more or less theoretical discussion may be supplemented by examples of ways in which such data may be applied in practical exploration, although the reader should keep in mind that the limitations of the data at present available prevent a comprehensive application.

Fields of Practical Application

One of the most useful aids to the successful exploitation of any orebody is an idea of the depth to which it may be expected to extend, and the ability to recognize the bottom of an orebody when it is reached is nearly as valuable as the ability to recognize the outcrop. Economic geologists have a pretty clear general idea of how deep oxidized orebodies and secondary (supergene) sulfide orebodies are likely to go under different conditions and what changes are likely to take place when these zones are passed. They also have some ideas as to what is to be expected in primary hypogene orebodies with increase in depth, but these ideas are not nearly as clearly defined and generally accepted as those relating to secondary ores. Every experienced mining man knows from observation that when he passes through the surface oxidized zone he will come into sulfides and that under normal conditions oxidized ore will not again be found beneath the sulfides. Likewise it is common knowledge that when the secondary sulfide zone has been passed ore of the same type will not be found again beneath primary ore. In regard to primary (hypogene) deposits, on the other hand, the relations of composition to depth are not so well understood as to afford much practical guidance to mining men. It has been customary to extend workings downward till funds give out or till the operator becomes discouraged by some inauspicious change in the orebody without determining conclusively whether or not the limit of profitable ore has been reached.

A more rational decision may be based on the nature of the change observed in the orebody. This change may show one of three aspects. First, there may be a narrow or lean zone in the oreshoot caused by structural conditions or a difference in the wall rock. Again, the vein or shoot may continue without change in size but undergo a gradual change in mineralogy and metal content. Thus a vein in which lead is an abundant constituent and copper subordinate may so change within a few hundred feet that lead is relatively subordinate and copper abundant, or a vein in which chalcopyrite is an abundant constituent may give place at depth to a vein consisting essentially of pyrite and containing very little chalcopyrite. Finally, there may be the more striking change —though one less often encountered—of a vein in which the metals are present as sulfides and allied minerals changing with depth to one in which much of the iron is present as oxide.

It seems clear that in the first case, where a vein decreases in content of a metal without notable change in mineralogy, there is a chance that the content of this metal may again increase at a deeper level, beyond the influence of the local structure, or change in the wall rock to which the change was due. On the other hand, where a vein not only decreases in metal content but alters radically in the character of the minerals present, these changes should be carefully considered before large expenditures are made for deeper development.

Rocky Mountain Region

The principle finds application in the southern Rocky Mountain region, where, in general, the primary ores of all the common metals, except iron, manganese, and gold, contain the metals as sulfides or allied minerals of the sulfide group. Any indication, then, of passage from the sulfide group of minerals to the oxides, spinels, and silicates is highly significant, and usually it is unfavorable. The most abundant metal that occurs in all these groups of minerals is iron, which, indeed, is practically everywhere present; and this metal is perhaps the best indicator of some critical change, because of its occurrence in different states of oxidation. Any locality where ferric minerals, whether oxides, spinels, or silicates, are present as vein minerals or are shown, by their presence unaltered in the wall rock, to have been in equilibrium with the ore solutions, lies either in the zone where sulfide could not form or on the border of the sulfide zone. In the former case no sulfides are likely to be found nearer the source of the solutions. In the latter case we may expect to find the sulfides increasing in one direction and decreasing in the other. This is perhaps the most general and certain rule that can now be stated; therefore the ferric iron content of the deposits in the wall rock, as well as the apparent stability of the ferric iron-bearing minerals, should be carefully watched.[9]

It is possible that a vein in which ferric minerals have been deposited may be reopened and sulfides be deposited, but in such veins it should be apparent that the ferric minerals were not stable and have been reduced by the later mineralizing solutions. The characteristic bleaching along sulfide veins of "red rocks" is a striking example of the instability of ferric oxides in the presence of solutions that deposit sulfides. If the ferric minerals are sufficiently abundant they may retard the reduction of oxidized solutions containing sulfur, and thus prevent or delay the normal deposition of sulfides, as has perhaps occurred during the formation of the native copper deposits of Michigan.

Ferric minerals are most abundant in contact deposits in limestone, probably owing to the oxidizing effects of carbon dioxide liberated from the limestone, but they are present in other deposits, and where they are not present in the vein their degree of stability may be noted in the wall rock.

Camp Bird Vein

A specific example of a problem in this field is afforded by the Camp Bird vein, in the Ouray district, Colorado. The writers have made a preliminary examination of this vein, and from the study of reports and discussion with those familiar with the deposit the conditions may be outlined essentially as follows. In the levels above the third adit, 600 to 1000 ft. below the surface, the vein is essentially a quartz-sulfide vein that had been repeatedly opened and mineralized.[10] There is no mention in the earlier reports of hematite or any ferric oxide minerals in the vein, although deeper explorations have encountered much hematite in the vein and much epidote in the altered wall rock. In the hematite shoot at the east end, the deepest shoot and the one last worked, part of the gold occurs in a quartz-hematite vein containing some sulfides. On the lower tunnel level, which is several hundred feet vertically below the main productive zone, the vein contains abundant specularite together with some sulfide, but the gold content is usually low. The question arises, is the specularite vein an earlier one that was reopened and mineralized with gold and sulfides, or has the quartz-sulfide vein of the upper levels gradually changed with depth to the specularite-bearing vein? The determination of such relations is of the highest importance in deciding the future work on such a vein. If the specularite vein is distinctly older than the sulfide vein, and if hematite has been destroyed by the latter mineralizing solutions, it may be that valuable shoots of ore exist at lower levels than the main productive shoots. If, however, the sulfide vein has gradually changed in depth to a specularite-bearing vein, with a loss of gold content, there is little likelihood that the sulfides and gold will reappear at still greater depths, at least within the range of the same type of wall rock.

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[9] This has already been pointed out by B. S. Butler: Suggested Explanation of the High Ferric Oxide Content of Limestone Contact Zones. *Econ. Geol.* (1923) **18,** 403.

[10] J. E. Spurr: The Camp Bird Compound Veindike. *Econ. Geol.* (1925) **20,** 115.

Colorado Bonanza District

Similar problems arise in the Bonanza district of Colorado, where during the early stages of mineralization hematite-bearing (specularite) silicified rocks or jaspers were formed. These hematite-bearing rocks were clearly unstable during the deposition of the later sulfides, the evidence of alteration being found in bleaching and pyritization of the jaspers with partial destruction of the ferric minerals. Along certain fault veins bodies of hematite-bearing rock may be found practically unaltered by sulfide-depositing solutions. Such veins are apparently unfavorable to the presence of commercial bodies of sulfides, either at the horizon, which is now the surface, or at greater depths. Other veins are characterized by marked pyritization and bleaching of the hematite-bearing rocks, and such veins evidently lie within a zone that would be favorable to the formation of sulfide orebodies if certain structural conditions, such as the presence of open spaces for the localization of sulfides in commercial quantities, were present.

Order of Deposition of Minerals.—In the table showing the stability fields of minerals and in the chart of electrode potentials it seems evident that the order of deposition of the mineral groups was silicates, oxides, and sulfides, the groups falling in the decreasing order of the average potentials of the elements constituting them. But there is also a suggestion that in any group of simple compounds, such as oxides or sulfides, the individual elements are deposited in the decreasing order of their potentials, though there are some prominent exceptions which need explanation.

From this consideration, molybdenum, zinc, and iron sulfides would be expected to form early in the sulfide stage; silver and mercury late in that stage, and lead and copper between these extremes. It has already been pointed out that a mineral containing a combination of two metals, such as iron and copper in chalcopyrite, actually forms between the stages where the simple sulfides would respectively form. These ideas as to the order of deposition within mineral groups are merely added for what they are worth as observations, without an attempt to explain certain anomalies which appear. This seems to be a field worthy of study, for so far as the writers are aware no satisfactory explanation has ever been advanced to account for the remarkable constancy of the order of deposition of minerals.

Problems also arise in connection with the state of oxidation of hypogene solutions, especially with reference to the possible high concentration of the strong acid radicals when sulfur forms mainly oxidized compounds. A discussion of these problems will not be attempted here.

It is hoped that with further study a more definite and comprehensive arrangement of the minerals as well as of metals may be determined, to assist in estimating positions in the veins relative to their changes with depth. This statement does not imply, however, that the writers wish to reserve this field of investigation to themselves.

SUMMARY

In the preceding pages the relation of the electrode potentials of elements to the occurrence of the elements in hypogene ore deposits is brought out. Electrode potentials are regarded by chemists as indicating certain fundamental properties of elements that have a direct and controlling influence on oxidation-reduction reactions. Such reactions are believed by the writers to have a vital bearing on ore deposition.

Arranged in the order of their electrode potentials, the groups of elements show a periodicity and a definite and striking relation to their occurrence in primary ore minerals. Elements in each group with electrode potentials above $+1$ occur only in oxygen combinations. Elements with negative potentials occur only as sulfides, allied minerals or native metals. Elements falling between the two extreme groups occur both as oxygen minerals and as sulfides.

In ore deposits the oxygen minerals of this intermediate group begin to form before the sulfides. The electrode potential of the sulfide ion S^{--} falls in the intermediate group, and under conditions that would cause ferrous iron to be oxidized to ferric iron, sulfide ion would probably be raised to a higher state of oxidation. The presence of barite and other sulfates formed before sulfides indicates that the sulfur is, in part at least, in a very high state of oxidation prior to the deposition of sulfides. As some of the metals were present as oxidized compounds before they were deposited as sulfides, it is questioned whether most of the metals were not transported in oxidized compounds rather than as soluble sulfides.

The groups of elements are deposited in the order of their potentials, and within the range of simple sulfides the metal sulfides tend to be deposited in the order of their potentials.

A further study of the problem may well give a much more definite order of mineral formation, with a logical chemical explanation which should add greatly to our understanding and certainty regarding the position of minerals in the general range of primary mineral formation.

Certain practical results are pointed out: If a vein contains ferric minerals that have not been attacked or altered by the ore solutions, the conditions were not favorable to the deposition of sulfides, and the sulfides are likely to decrease toward the source of the solutions. Among the sulfide minerals there is an order of deposition from higher to lower temperature that corresponds with the electrode potentials of the metals.

ACKNOWLEDGMENTS

The writers acknowledge the helpful criticism of several chemists, including Dr. F. M. Coolbaugh and Prof. T. P. Campbell, of the Colorado School of Mines; John Sullivan, of the United States Bureau of Mines Station at Tucson, Ariz.; Dr. E. T. Allen, of the Geophysical Laboratory; and Dr. R. C. Wells, of the United States Geological Survey; also of their geologic associates, especially their colleague in the Colorado work, Dr. T. S. Lovering.

DISCUSSION

H. R. BLANK, New York, N. Y. (written discussion).—The electrode potentials of the elements have been measured between the elements and aqueous solutions of their ions, and have been shown to apply to oxidation-reduction reactions in aqueous solutions. Would the same values of the electrode potentials apply to the prediction of reactions in fused solutions?

R. J. LEONARD, Tucson, Ariz. (written discussion).—Electrochemical action as a frequent factor in ore deposition is generally recognized by students of ore deposits, though perhaps not always to the extent it evidently deserves. The authors' arrangement of tables and the arguments put forth certainly indicate relationships of a nature as yet unconsidered by investigators. Moreover, the laboratory work of this kind that has been done by Gottschalk and Buehler, Wells and others strongly suggests that further and perhaps more definite results are yet to be obtained for geological application, when experimental investigations are carried out with this particular end in view. Of course, laboratory work bearing on the subject of ore deposition in general postulates rather ideal conditions of process, whereas in nature many unknown conditions doubtless exist to produce quite similar results—or different, as the case may be; therefore most results obtained experimentally are of doubtful or, at best, of limited practical value. This may be more than usually true of research dealing with electric activity.

In any attempt to correlate the electrode potentials of the chemical elements with the position and mode of combination of the elements as found in ore deposits, various important and often conflicting factors enter into the problem, as even a brief inquiry into electrochemical phenomena will show. Nevertheless, there are not so many known conditions but that a sufficiently large number of experiments in various combinations may provide quite enough definite, usable data to be of geologic value.

It is hoped that the present paper will stimulate interest and experimental activity in this phase of the big and important problem of ore deposition.

T. P. Campbell, Golden, Colo. (written discussion).—The appearance of this paper is most welcome, as it displays the increasing tendency toward the use of the methods of physical chemistry in dealing with seemingly complex and esoteric natural phenomena. The discussion that follows is not intended as a criticism, but rather as an amplification of the main points brought out in the paper.

First a few definitions: From the physicochemical standpoint, "oxidation" means loss of electrons; "potential" means the work required to transport a unit electrical charge through a given length of circuit under given conditions. If the circuit contains a metal/liquid junction, the quantity of electricity is expressed in terms of the Faraday (96,500 coulombs). Hence, when we speak of the "oxidation potential" or of the "standard electrode potential" of, say, zinc, what we really mean is the work required to transport two Faradays of electricity *to* the metal zinc *from* one gram-molecular weight of zinc ions, Zn^{++}, contained in 1000 g. of water at given temperature and pressure. Or, which comes to the same thing, this represents the energy that is *liberated* when one mole of zinc loses two Faradays (as electrons). Thus at 25°C., E_0 for Zn^{++} is given as -0.758 volt; then the energy given up to the surroundings by the reaction

$$Zn + 2E^- \rightarrow Zn^{++}$$

must be

$$\triangle F = \frac{-0.758 \times 2 \times 96{,}500}{4.19} = -34{,}980 \text{ cal. per mole,}$$

which is, of course, the "free energy of formation" of zinc ion at 25°C.[11]

Thus there is a definite relationship between potential and free energy, and the terms may be used interchangeably if the implications involved are fully appreciated. As a rule, however, there seems to be considerable confusion regarding the true significance of the terms "oxidation," "potential," "reduction," and "affinity"—or "antipathy;" hence it is suggested that the use of free energy as a measure of affinity, or lack thereof, should lead to a clearer conception of the essential simplicity of predicting, or accounting for, the probable course of physicochemical reactions.

We are wont to account for reactions by assuming the action of a *force*, usually external to the system. If we think of a *force* as the tendency of energy to flow from a region of high intensity to one of low intensity, the essential role of energy flow in all natural phenomena becomes apparent; and the value of the concept of affinity in terms of free energy permits us to apply the well-known laws of thermodynamics to the problem of the formation of hypogene deposits.

Thus we need not assume any particular mechanism or type of solution if we can determine the quantity and direction of the energy flow involved. In the first place, we may note[12] that most of the common cations and anions represent a lower energy level than do the corresponding elements either in mixture or in atomic or molecular solution (dispersion). Hence it is apparent that the solutions from which the primary deposits resulted contained the metals, non-metals, and gases, for the most part as ions.

Second, it is reasonable to suppose that, at depth, these solutions existed under considerable pressure and relatively high temperatures, and that as the solutions approached the surface the temperature and pressure both decreased progressively. Since aqueous solutions are comparatively noncompressible, it is evident that the main effect of high pressure on such solutions would be to prevent the escape of the more volatile constituents—such as O_2, CO_2, H_2S, SO_2, etc.—thus giving rise to reactions and equilibria not attainable at ordinary pressures. The effect of temperature change, however, is much more profound, since the value of the free energy change, and hence of the affinity, may be greatly modified, or even reversed in sign at high temperatures.[13] For example, at 25°C. the reaction

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[11] For the sake of uniformity, the notation used in Lewis and Randall's Thermodynamics [New York, 1923, McGraw-Hill Book Co.] will be used here. For standard molal potentials see p. 420; for standard free energies, see p. 607.

[12] G. L. Lewis and M. Randall: *Op. cit.*, 607.

$$SO_2 + \frac{1}{2}O_2 \rightarrow SO_3$$

gives up to the surroundings 85,890 cal. of free energy; but at 787°C. the free energy change is zero, and above this temperature the reaction (if forced) absorbs energy.

Third, we come to the question of solubility, or of "solubility product," as it is sometimes called. This is a moot point, but we may note that the solubility of a given substance is again a question of the free energy change, or of energy flow. In general, if a compound is formed and precipitated from solution, we may assume that the cause is to be found in a liberation of energy; that is, in such a case the solid state in contact with the solution (heterogeneous system) represents a lower level of energy than did the mother solution (homogeneous system).

TABLE 2.—*Heats of Formation*

(Gram Calories per Mole)

Metal M^{++}	Silicates $MO.SiO_2$	Sulfates MSO_4	Carbonates MCO_3	Oxides MO	Sulfides MS	Selenides MSe[a]
Ba	328,100	339,400	286,300	133,400	102,900	69,900
Na	326,000	328,100	273,700	100,900	89,300	60,900
Ca	329,950	317,400	273,850	131,500	94,300	58,000
Zn		229,600	197,500	84,800	43,000	30,000
Cd		219,900	183,200	66,300	34,400	23,700
$Cu^{\cdot\cdot}$		181,700	146,100	37,700	10,100	17,300
Pb		215,700	170,000	50,800	20,200	17,000
$Fe^{\cdot\cdot}$	254,600		187,800	65,700	24,000	15,700
$Cu^{\cdot}$				43,800	20,300	8,000
$Hg^{\cdot\cdot}$		165,100		21,500	10,600	6,300
Ag		167,100	123,800	7,000	3,000	2,000

[a] The tellurides are of the same approximate order of magnitude.

To take up the more specific problem: Assuming that solutions at depth contain a number of possible ionic constituents, what are the most probable combinations of these ions and at what depths, or in what order, will such combinations be most likely to occur? A quantitative answer to the first question could be given if we knew all of the energy values involved, but unfortunately such data are largely lacking. However, we may use heats of formation as an approximation, where free energy values are lacking. A few comparative values of heats of formation are given in Table 2,

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[13] G. L. Lewis and M. Randall: *Op. cit.*, Chap. IX and XXXI. A simple way of showing the affect of temperature and of ion concentration is by means of the Nernst equation

$$\Delta F = \Delta F_o + RT\ln(C_m),$$

where C_m means the molal concentration of the given ion. This also brings out the relatively small affect of concentration changes.

to bring out the application of such data.[14] In this connection it may be mentioned that Thomson's rule may be used to calculate the corresponding electrode potential, but that this potential differs from the standard value by the amount of the temperature coefficient, $T(dE/dT)$.

Thus, since at depth the escape of volatiles is relatively unlikely, and from the energy values involved, the order of deposition of compounds of a given metal would probably be: (1) silicate, (2) sulfate, (3) spinel, (4) carbonate, (5) oxide, (6) sulfide, (7) selenide.

It may be noted also that the more basic (electronegative) the metal, the higher it would stand in any group in this series. In this connection it is interesting to note the high stability indicated for the more basic and complex silicates; for example, as compared to the simple $CaSiO_3$ value given above, the corresponding values for other silicates are

	Cal. per Mole
Ca_2SiO_4 =	471,300
Ca_3SiO_5 =	603,000
$Ca_3Al_2Si_2O_{10}$ =	1,195,400
$H_4Al_2Si_2O_9$ =	927,400

The case of that ubiquitous element, iron, is interesting as showing the range of distribution possible for a polyvalent, weakly basic element that exists in large total amount throughout the lithosphere. Halferdahl gives the heat of formation of FeS_2 as 35,500 cal. per mole,[15] and notes that on heating at ordinary pressures

$$7FeS_2 \rightarrow Fe_7S_8 + 3S_2 + 3(9880) \text{ cal.}$$

In other words, FeS_2 appears to be a condensation product, rather unstable at higher temperatures unless a relatively high temperature be maintained. Again, Lewis and Randall[16] give the free energy of formation of Fe^{++} as $-20,350$ cal. (slightly greater than that for Cd^{++}), while that of Fe^{+++} is only $-3,100$ cal. These facts, taken in conjunction with the fact that energy must be pumped into the reaction

$$S^0 \longrightarrow S^{--} - 2E^-$$

in molal solution, indicates that the "antipathy" of ferric iron to sulfides is simply a matter of the relatively small amount of energy available in Fe^{+++} as compared to that in Fe^{++} or in other more electronegative metals. That is, ferric iron would be more "attracted" to the oxidized compounds where the free energy decrease is high.

Finally, and in summary, if we regard these mineral-bearing solutions simply as physicochemical systems, and apply to them even the limited amount of data now available, in the light of the known laws of thermodynamics, we arrive at a semiquantitative answer to the problem of the relation of composition to depth. As the authors state, " . . . the individual elements are deposited in the decreasing order of their potentials, though there are some prominent exceptions which need explanation." It is probable that these exceptions can be brought into line by taking account of the effects of temperature and ion concentration, as indicated above. And in any event, the application of the general principles established cannot but prove fruitful when carefully and critically interpreted.

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[14] See H. O. Hofman: General Metallurgy, chap. 4. New York, 1913, McGraw-Hill Book Co.; also Landoldt-Börnstein Tabellen, etc.

[15] A. C. Halferdahl: Thermal Decomposition of Covellite and Pyrite. *Min. & Met.* (1927), 8, 469.

[16] G. L. Lewis and M. Randall: *Loc. cit.*, 607.

A. I. M. E. Technical Publications, 1929

[Separates of all the *Technical Publications* published in 1929 are available at Institute headquarters. All the papers are on file in public, university and technical libraries, and when so indicated in the list, may be found in the Transactions.]

Metal Mining

Technical Publication No.	Title	Author	Meeting	Trans. Volume and Page*
182-A-23	The Royal Commission on Mining Subsidence	Henry Louis	New York Feb., 1929	
183-A-24	Cycles in Metal Production	D. F. Hewett	New York Feb., 1929	**T29**†, 65, *93*
199-A-25	Ventilation at the United Verde Mine	Oscar A. Glaeser	New York Feb., 1929	**T29**, 114, *141*
206-A-26	Air Conditioning in Deep Mines	R. W. Waterfill	New York Feb., 1929	**T29**, 99, *112*
245-A-27	Bumps in No. 2 Mine, Springhill, Nova Scotia (with discussion)	Walter Herd	New York Feb., 1929	
257-A-28	Misfires: Their Causes, Prevention and Treatment on Occurrence		New York Feb., 1929	
	Misfires in Anthracite Coal Mines (With Discussion)	T. D. Thomas		
	Misfires in Bituminous Coal Mines (With Discussion)	W. H. Forbes		
	Misfires in Non-metallic Mining (Limestone) (With Discussion)	A. W. Worthington		

Milling and Concentration

Technical Publication No.	Title	Author	Meeting	Trans. Volume and Page*
195-B-15	A Study of Differential Flotation	C. R. Ince	New York Feb., 1929	
196-B-16	Elements of Operation of the Pneumatic Table	Arthur F. Taggart R. L. Lechmere-Oertel	New York Feb., 1929	
202-B-18	Silver-bearing Minerals of Some Ores from Tintic Mining District	A. W. Hahn	New York Feb., 1929	**T29**, 325
204-B-17	Experiments with Flotation Reagents	A. F. Taggart T. C. Taylor C. R. Ince	New York Feb., 1929	
208-B-20	Cyanide Regeneration or Recovery as Practiced by the Compania Beneficiadora de Pachuca, Mexico	C. W. Lawr	San Francisco Oct., 1929	
214-B-19	Calculations in Ore Dressing	W. Luyken E. Bierbrauer	New York Feb., 1929	
217-B-21	Differential Grinding Applied to Tailing Retreatment	Leon M. Banks George A. Johnson	San Francisco Oct., 1929	
224-B-22	Reducing and Oxidizing Agents and Lime Consumption in Flotation Pulp	Research Staff of United Verde Copper Co.	San Francisco Oct., 1929	
247-B-23	Copper Sulfate as Flotation Activator for Sphalerite	O. C. Ralston C. R. King F. X. Tartaron	San Francisco Oct., 1929	

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* Volume numbers are in bold face; page numbers of papers, Roman; and page numbers of discussions, italics. Where no number is given, paper has not appeared in Transactions. Discussions are printed following the respective paper.

† **T29** indicates inclusion in general volume Transactions (1929).

MILLING AND CONCENTRATION.—*Continued*

TECHNICAL PUBLICATION No.	TITLE	AUTHOR	MEETING	TRANS. VOLUME AND PAGE
248-B-24	Activation of Sphalerite for Flotation	Oliver C. Ralston William C. Hunter	San Francisco Oct., 1929	
250-B-25	Effect of Copper and Zinc in Cyanidation with Sulfide-acid Precipitation	E. S. Leaver J. A. Woolf	San Francisco Oct., 1929	
225-B-26	Microscopic Studies of Mill Products as an Aid to Operation at the Utah Copper Mills	H. S. Martin	San Francisco Oct., 1929	

IRON AND STEEL

TECHNICAL PUBLICATION No.	TITLE	AUTHOR	MEETING	TRANS. VOLUME AND PAGE
159-C-21	Coal Washability Tests as a Guide to the Economic Limit of Coal Washing	George Stanley Scott	New York Feb., 1929	
162-C-20	Metallic Electrodes for Cast-iron Arc Welding	Shun-ichi Satoh	New York Feb., 1929	C29*, 144, *158*
165-C-22	Unreduced Oxides in Pig Iron and Their Elimination in the Basic Open-hearth Furnace	C. H. Herty, Jr. J. M. Gaines, Jr.	New York Feb., 1929	C29, 179
168-C-23	Gases Extracted from Iron-carbon Alloys by Vacuum Melting	N. A. Ziegler	New York Feb., 1929	C29, 428, *440*
169-C-24	The Gamma-alpha Transformation in Pure Iron	Albert Sauveur C. H. Chou	New York Feb., 1929	C29, 350, *361*
170-C-25	The Blast-furnace Theory	Richard Franchot	New York Feb., 1929	
181-C-26	Effect of Barium Oxide on the Desulfurizing Power of Blast-furnace Slags	C. E. Wood T. L. Joseph	New York Feb., 1929	C29, 126, *142*
183-C-27	Cycles in Metal Production	D. F. Hewett	New York Feb., 1929	T29, 65, *93*
184-C-28	A Metallographic Study of Tungsten Carbide Alloys	J. L. Gregg C. W. Küttner	New York Feb., 1929	E29†, 581, *590*
185-C-29	Pure Iron and Allotropic Transformations	T. D. Yensen	New York Feb., 1929	C29, 320, *333*
186-C-30	Basic Open-hearth Yields	C. D. King	New York Feb., 1929	C29, 197, *232*
193-C-31	Relation of Nitrogen to Blue Heat Phenomena in Iron and Dispersion Hardening in the System Iron-nitrogen	R. S. Dean R. O. Day J. L. Gregg	New York Feb., 1929	C29, 446
205-C-32	The Passivity of Metals and Its Relation to Problems of Corrosion	Ulick R. Evans	New York Feb., 1929	E29, 7
218-C-33	Change in Microstructure of Iron at A_3 Transformation Point	B. A. Rogers	Cleveland Sept., 1929	C29, 370, *380*
227-C-34	Iron-ore Sinter	G. M. Schwartz	Cleveland Sept., 1929	C29, 39, *66*
228-C-35	A New Development in Wrought Iron Manufacture	James Aston	Cleveland Sept., 1929	C29, 166, *176*
229-C-36	Diffusion of Iron Oxide from Slag to Metal in the Open-hearth Process	C. H. Herty, Jr.	Cleveland Sept., 1929	C29, 284, *299*
230-C-37	Experimental Data on the Equilibrium of the System Iron Oxide-carbon in Molten Iron	A. B. Kinzel J. J. Egan	Cleveland Sept., 1929	C29, 304, *311*
249-C-38	Blast-furnace Filling and Size Segregation	C. C. Furnas T. L. Joseph	Cleveland Sept., 1929	C29, 98, *124*

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* C29 indicates inclusion in TRANSACTIONS, Iron and Steel Division, A. I. M. E. (1929).

† E29 indicates inclusion in TRANSACTIONS, Institute of Metals Division, A. I. M. E. (1929).

IRON AND STEEL.—*Continued*

TECHNICAL PUBLICATION No.	TITLE	AUTHOR	MEETING	TRANS. VOLUME AND PAGE
260-C-39	Influence of Nitrogen on Special Steels and Some Experiments on Case-hardening with Nitrogen	Shun-ichi Satoh	New York Feb., 1930	
261-C-40	Reclaiming Steel-foundry Sands	A. H. Dierker	New York Feb., 1930	

NON-FERROUS METALLURGY

TECHNICAL PUBLICATION No.	TITLE	AUTHOR	MEETING	TRANS. VOLUME AND PAGE
209-D-15	Blast Roasting at Cerro de Pasco	Glenn A. Keep	New York Feb., 1929	**T29**, 230, *239*
220-D-16	Formation of Insoluble Zinc Compounds during Roasting (With Discussion)	H. R. Hanley C. Y. Clayton David Walsh	New York Feb., 1929	**T29**, 215, *227*
221-D-17	Lead Alloys for Anodes in Electrolytic Production of Zinc of High Purity (With Discussion)	U. C. Tainton A. G. Taylor H. P. Ehrlinger	New York Feb., 1929	**T29**, 192, *200*
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223-D-19	Handling of Fine Ores and Concentrates in Salt Lake Valley Lead Smelters (With Discussion)	L. D. Anderson	New York Feb., 1929	**T29**, 179, *186*
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